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Canada Centre for Mineral and Energy Technology | Centre canadien de la technologie des minéraux et de l'énergie

MODELLING OF 830 OREBODY OF COPPER CLIFF SOUTH MINE, INCO: PART 1 - A PRELIMINARY ASSESSMENT USING THE MINTAB PROGRAM

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MINING RESEARCH LABORATORIES

AUGUST 1981

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by

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1. SUMMARY

A cooperative study was recently initiated between INCO Metals Company, Sudbury Operations and the Mining Research Laboratories of CANMET, E.M.R. The primary objectives are (a) to predict stresses and deformational changes around excavations during mining and pillar recovery at the 830 orebody of Copper Cliff South (C.C.S.) mine, and (b) to assess the validity and applicability of numerical modelling in predicting ground behaviour in response to mining and backfill operations at great depth.

To provide realistic input data for the numerical model, studies will be carried out to determine primitive stresses, rock properties, stope convergence and stress changes induced by mining in pillars, fill,....etc. Some of the measurements will also be used to verify the degree to which simulation predicts actual mine performance.

This report describes the preliminary trial of modelling 830 orebody of C.C.S. mine using the displacement discontinuity technique. Pillar compressibility is taken into account but backfill is not simulated at this stage. All input parameters for modelling were based on data derived from other mines in the Sudbury area. Preliminary results seem to be very reasonable and encouraging.

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2. INTRODUCTION

The Canadian mining industry, in order to maintain its competitive position and traditional leading role in mining innovation, needs to develop new techniques which can be applied to the mining of large, low grade base metal deposits at depth. To achieve this goal considerable research effort is required to improve existing mining methods, support systems, and backfill procedures and to develop computer-aided design tools for mine designs in order to achieve better ground control.

Blasthole open stoping with delayed backfill has the potential of providing higher productivity at lower mining costs per ton of ore. In addition it reduces the need for miners to work inside stopes and as a result avoides potential accidents from falls of ground. It also eliminates the need for cyclic support installations in stopes.

INCO's Copper Cliff South mine is currently mining 830 orebody with blasthole stopping. Mining of the orebody is presently in the primary stoping phase. This will provide an opportunity for INCO Metals Ltd and the Mining Research Laboratories to establish a case history of ground behaviour for open stoping with delayed backfill. The ultimate aim is to develop and to test the predictive capability of numerical modelling when applied to ground behaviour in case of open stoping with delayed backfill. If the modelling approach and investigational method used in this joint project proves to be satisfactory, then the experience gained from this study would be useful for mining operations of a similar type.

3. MODELLING TECHNIQUES

Several numerical methods have been examined in connection with this project. Due to the complexity of mining geometry and the extent of the area of the mine involved, this mining operation does not lend itself to two-dimensional simulation. Therefore simulation using well-developed two-dimensional finite element (F.E.) techniques was ruled out.

No attempt was made to use available three-dimensional (3D) F.E. techniques to simulate the 830 orebody although they have been developed for mining and geotechnical applications (1). The use of three dimensional F.E. real scale modelling of the 830 orebody taking into account realistic rock mass behaviour would be too expensive.

The possibility of using three dimensional linear, elastic boundary integral (boundary element) method was also examined. It was concluded that it was impractical for simulating such a large mining area.

The displacement discontinuity method on the other hand seems to be very flexible and applicable for modelling this type of mining geometry. The method is of interest because it was developed from procedures designed especially to handle mining problems in which excavations were treated as slits in the rock medium (2, 3). The analytical technique is applicable to extraction of plane deposits which may have any orientation relative to ground surface. It is assumed that the mine workings are sufficiently deep so that the influence of the surface on stresses and displacements near the excavations can be neglected. It is also assumed that the plane of excavation is uniformly inclined, uninterrupted by faults and dykes and consists of excavations of uniform stoping width. Details of the mathematical formulation of the diplacement discontinuity method are given elsewhere (2).

This method is capable of handling complex stope-and-pillar geometries in the mining horizon; extraction sequences and backfill operations can be easily simulated. In addition, it is relatively easy to prepare the input data and it is inexpensive to run as compared to other method such as the F.E. However this simulation technique also requires some knowledge of the elastic behaviour of the orebody and its surrounding rocks. Note that in this preliminary study the backfill option was not considered.

With these limitations and realizing the fact that the 830 orebody of C.C.S. mine is relatively thick, the model should provide a first order approximation to stresses and displacements around the excavations in the mining horizon. If this is established then this numerical technique will be a useful tool for mining engineers in making design decisions and selecting mining sequences at great depth.

The computer program used for this study is the MINTAB code which was originally obtained from Consolidated Goldfields Corporation of South Africa and has been extensively modified at the Mining Research Laboratories (3, 4, 5). This program has been used very satifactorily in connection with the study of regional stability of uranium mines in Elliot Lake, Ontario.

4. BRIEF DESCRIPTION OF COPPER CLIFF SOUTH MINE

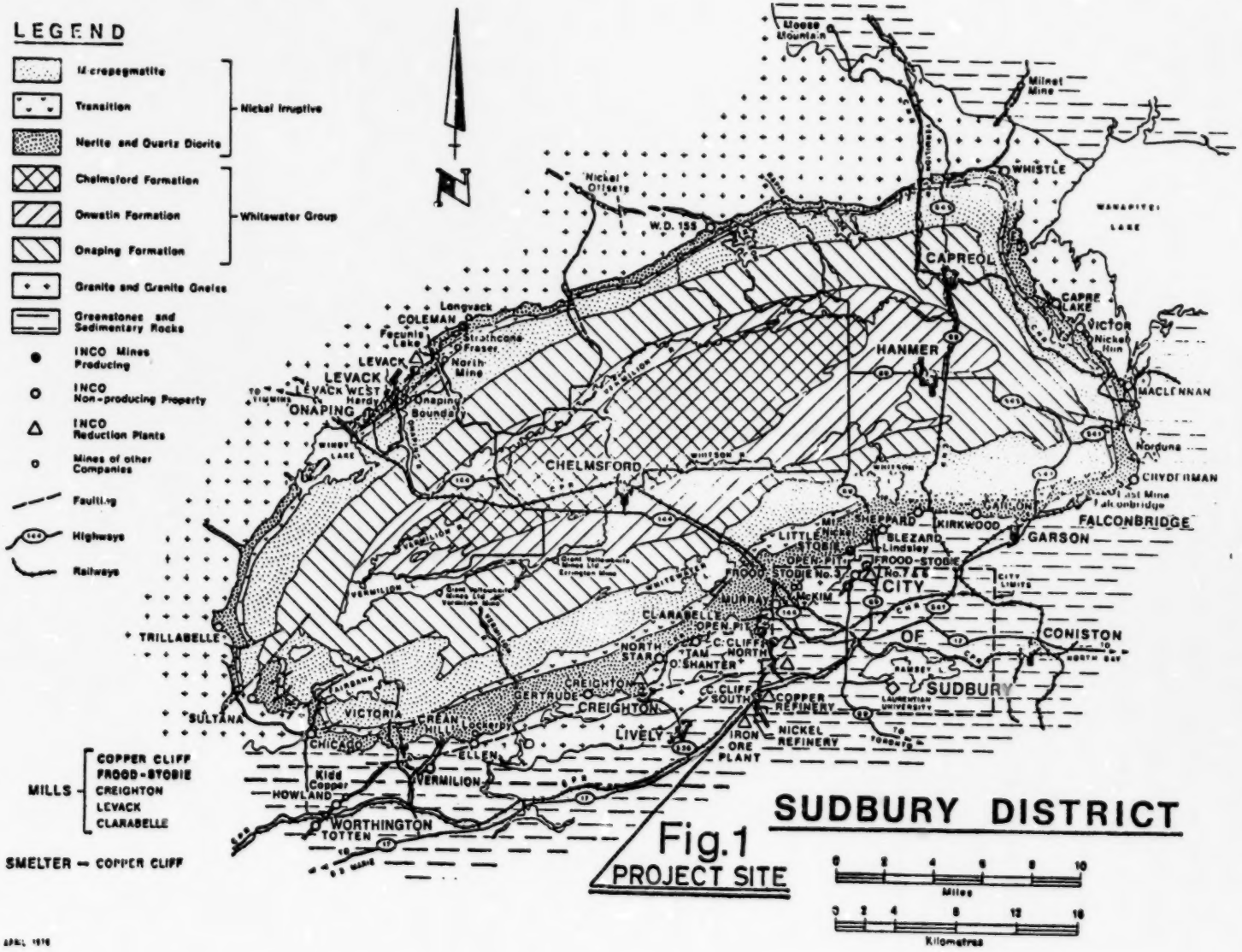
The location of Copper Cliff South mine is shown in Fig. 1 and a brief description of the operation is given below based on a tour description given by the CIM Reporter (6):

"Mining operations at South Mine began with the sinking of No. 1 and No. 2 shafts, in 1967, by a contractor, to obtain access to the main orebody. No. 2 shaft was completed in 1969 to a depth of 2300 feet and then converted to a fresh air raise in 1972. No. 1 shaft was completed in 1972 to a depth of 4245 feet.

The early mining methods were shrinkage and LHD blasthole mining. Cut-and-fill mining was started in 1976 with the completion of the sandplant construction in late 1975.

The average daily production, in 1981, is scheduled at 3555 tons per day and will be realized through the following mining methods: in-the-hole (6.5 in. diameter) blasthole @ 23 per cent, in-the-hole (6.5 in. diameter) vertical retreat mining @ 40 per cent, cut-and-fill mining 34 per cent and, finally, development @3 per cent.

All development horizons, except one, are advanced trackless, using scooptrams, drill jumbos, including two Montabert electric-hydraulic machines, scissor platform vehicles and haulage trucks. Raise development is achieved with standard methods as well as raise borers. At this time, a development program is in progress, with emphasis on th 810 orebody below 2250 level and the 865 orebody, located under the town of Copper Cliff.


LEGEND
Micropegmatite
Transition
Norite and Quartz Diorite
Nickel Irruptive
Chelmsford Formation
Onwatin Formation
Onaping Formation
Whitewater Group
Granite and Granite Gneiss
Greenstones and Sedimentary Rocks
INCO Mines Producing
INCO Non-producing Property
INCO Reduction Plants
Mines of other Companies
Faulting
Highways
Railways
MILLS
COPPER CLIFF
FROOD-STOBIE
CREIGHTON
LEVACK
CLARABELLE
SMELTER — COPPER CLIFF
APRIL 1976
Moose Mountain
Milnet Mine
WHISTLE
Nickel Offsets
W.D. 155
CAPREOL
HANMER
CHELMSFORD
ONAPING
LEVACK
COLEMAN
Longvack
Strathcona
Fraser
North Mine
Onaping Boundary
MACLENNAN
Norduna
CHRYSTMAN
FALCONBRIDGE
GARSON
KIRKWOOD
SHEPPARD
BLEZARD
Lindsley
FROOD-STOBIE No. 3
OPEN PIT
CITY OF SUDBURY
CONISTON
McKIM
MURRAY
CLARABELLE OPEN PIT
NORTH STAR
TAM O'SHANTER
C. CLIFF NORTH
C. CLIFF SOUTH
COPPER REFINERY
NICKEL REFINERY
IRON ORE PLANT
CREIGHTON
GERTRUDE
LIVELY
TRILLABELLE
SULTANA
VICTORIA
CHICAGO
ELLEN
VERMILION
Kidd Copper
HOWLAND
WORTHINGTON
TOTTEN
Miles
Kilometres
Fig.1
PROJECT SITE
SUDBURY DISTRICT

On 2050 level, two operations, development and tramming, are dependent on track. Production re-tramming of ore and removal of rock from near orebodies is achieved by means of a Grangesberg haulage system. The extension of this system to the 865 orebody is realized with a combination Haggloader and Shuttletrain, again dependent on track.

Most production blasting, in blasthole and vertical retreat mining, makes use of "multi-point initiation" through the application of the slider technique.

Final recovery of the in-stope ore, beyond the drawpoint brow, is realized through the use of remote, radio-controlled scooptrams.

Extensive use is being made of cement-grouted cable bolts to support crown pillars above mined-out blasthole stopes or in areas where adverse structures affect the strength of ground."

830 orebody strikes N-S and dips 65° W. It has a horizontal extension of 365 m (1200 lt) and a vertical extension of 275 m (900 ft). Figures 2 and 3 are cross-cut and longitudal sections respectively indicating present mining geometry in the vicinity of 830 orebody.

5. C.C.S. MINE MODEL AND INPUT PARAMETERS

Any theoretical analysis necessarily requires certain basic input data. The essential data for the present study with the modelling technique being used are:

a) The state of stress prior to mining.

b) The elastic reponse of the rock mass as characterized by its modulus of deformation and Poisson's ratio.

c) The deformational behaviour or stiffness of the pillar and/or backfill material if the compressibility of pillars and/or the effects of backfill are taken into account.

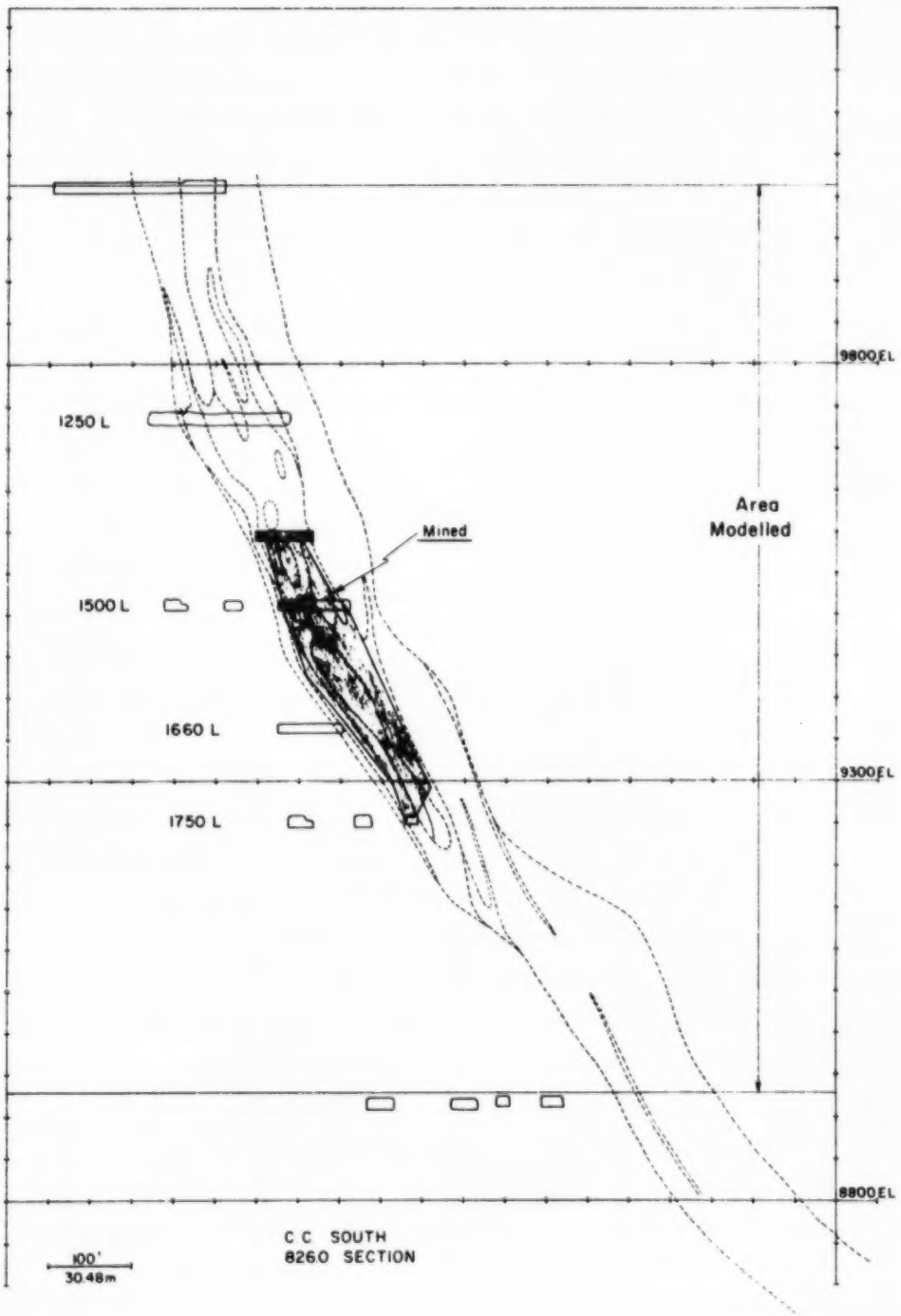


Fig. 2 - Cross section of C.C.S. Mine showing 826.0 section

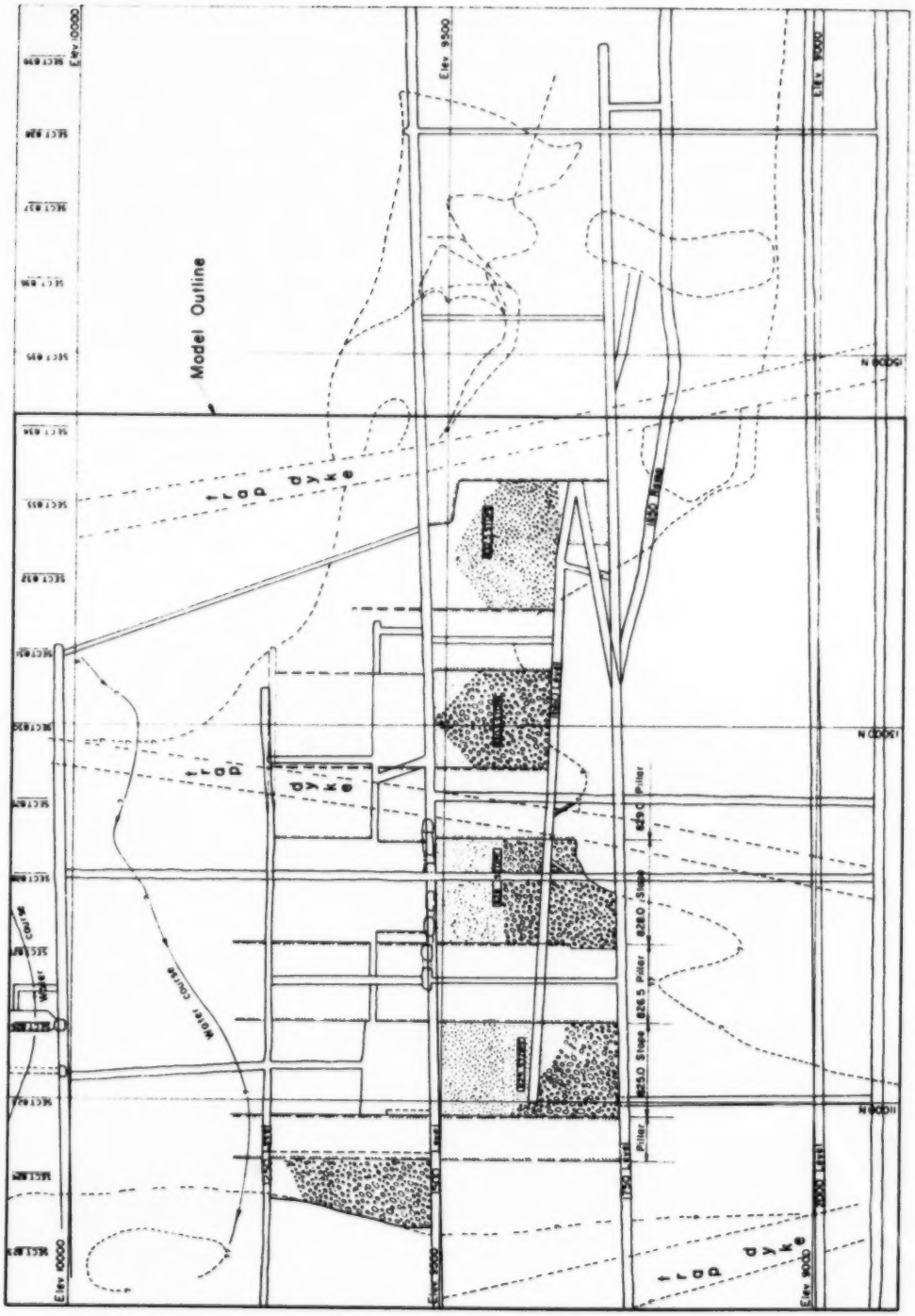


Fig. 3 - Longitudinal section of C.C.S. mine

None of these data is available at present for the test site; rock property testing and in situ stress determinations are planned or underway. However, to gain some experience on the use of this technique for modelling blasthole stoping operations, the following assumed properties based on information collected from the Sudbury area were used for this preliminary trial.

Modulus of deformation - H/W and F/W	6.9×10^4 Mpa
Poisson's ratio - H/W and F/W	0.25
Normal stiffness of pillars	6.9×10^4 Mpa
Sheer stiffness of pillars	2.76×10^4 Mpa
Unit weight of rock masses	0.026 MN/m^3
Ratio of pre-mining stress σ_{xx}/σ_{zz}	1.52
Ratio of pre-mining stress σ_{yy}/σ_{zz}	1.68
Where σ_{zz} = unit weigth of rock masses * depth	
Dip angle of reef plane	65^o
Stoping thickness	20m
Percent of primitive stress for convergence criterion.....	5

The general mining depth is 500 m. The first and the last row of the mine grid are at depths of 300 m and 631 m respectively. There are two mining sequences for the preliminary trial: step 1 - existing stoping and step 2 - completion of the primary stoping.

Because all the stopes and pillars have different widths with no common denominator a small element (square) size of 4.573 m has been chosen which yields a reasonable 365 m x 365 m size model. The mine model is shown in Fig. 4.

6. PRESENTATION OF RESULTS

The output from the MINTAB program consists of the following:

(a) Closure and ride components (DZ, DX, DY)

(b) The six stress quantities (SIGXX, SIGYY, SIGZZ, SIGXY, SIGYZ AND SIGZX) (a) and (b) are all expressed in the seam coordinate system.

(c) Released energy (ENERGY), and

(d) Stresses and displacements, in both seam and global coordinates systems, at any point within the rock mass.

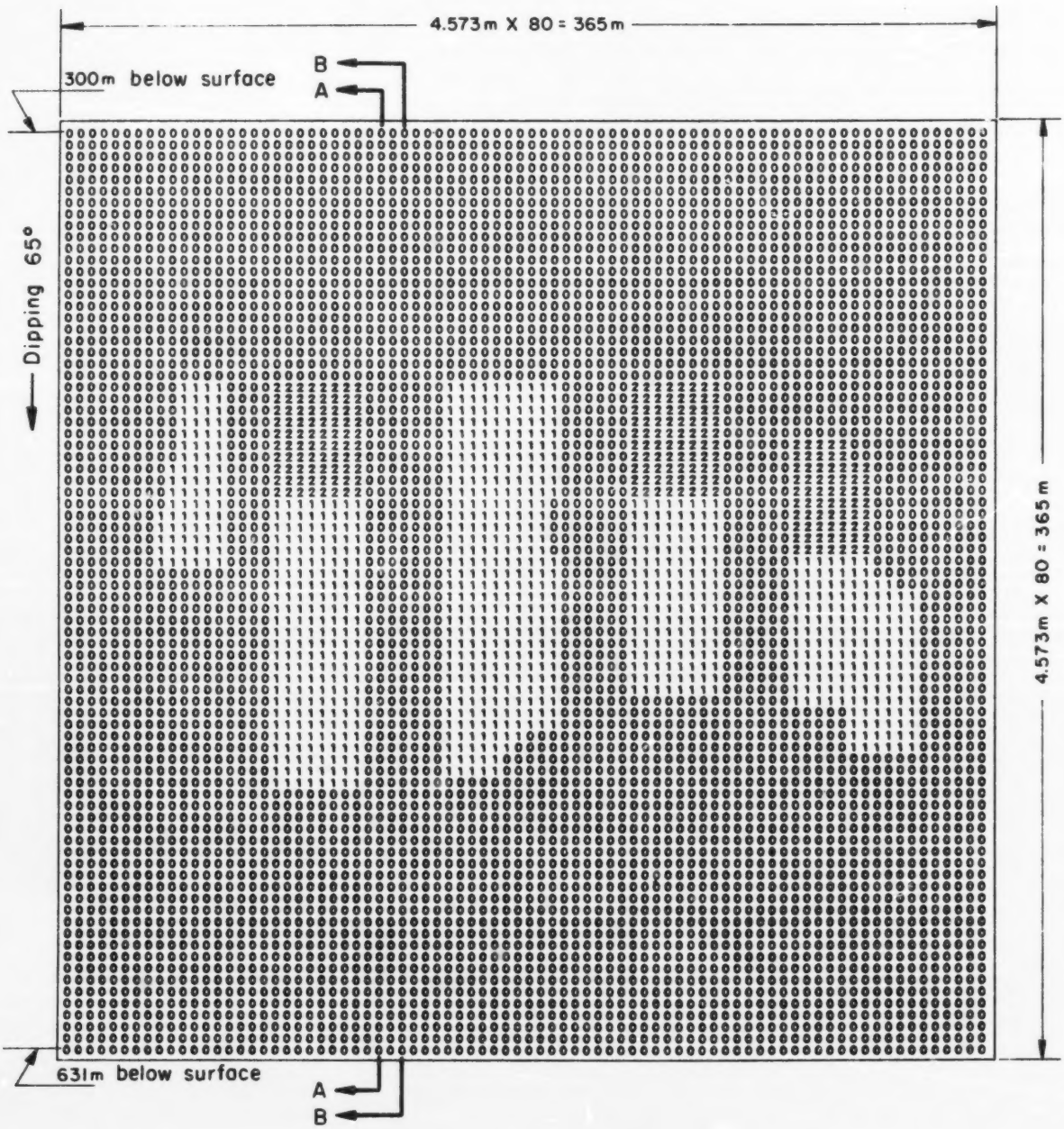


Fig. 4 - C.C.S. mine model

The printouts can be very selective and flexible. A control mechanism for printing results has been introduced using the large square (block) as a basic unit. In addition, a printer plot routine was designed and incorporated to represent stresses, displacement (closure/rides), released energy and strength/stress ratio, etc., in a form of point contours. In other words, these quantities are represented by symbols which display a range of stresses or closure/rides etc., rather than their absolute values.

6.1 POINT CONTOUR PLOTS

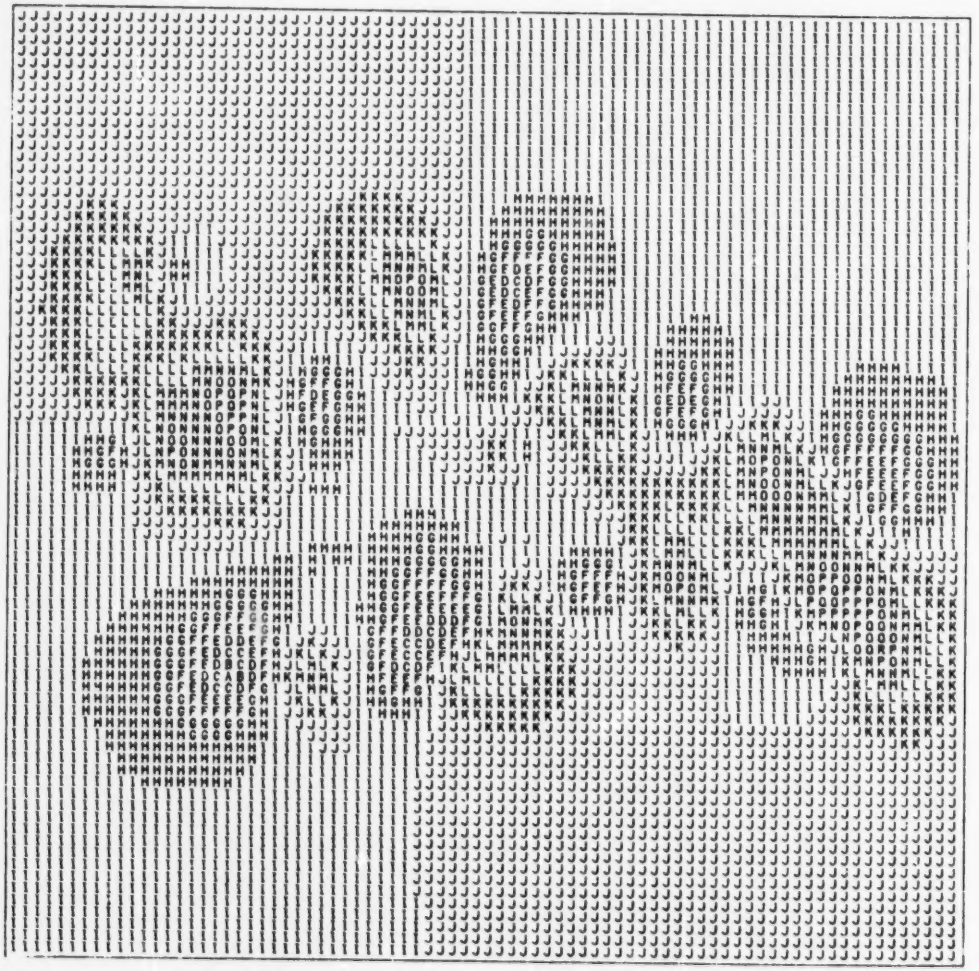
Any or all of the quantities i.e., rides/closure (DX, DY, DZ) stresses (SIGXX, SIGYY.....), released energy (ENERGY) and the strength/stress ratio (S.S.R.) can be printed along with other selected output using the print characteristics of a line printer. (Note that the released energy will not be calculated and printed for the first mining step).

These printer-plots provide a quick means to examine the response of rock mass to stresses induced by mining. However, these printer-plots do not yield a true scale representation of mine geometry, i.e., the scaling in the -x and -y directions (along the strike and dip) are different.

To overcome the shortcomming of line printer-plotting a separate plotting routine has been developed for the Calcomp plotter. Results from MINTAB program are usually stored on a disc or tape file for this purpose. All plots presented in this report were done on the Calcomp plotter.

These point contours are shown in Figs. 5 to 14 and Figs. 15 to 25, respectively, for mining steps 1 and 2. Among the plots, the point contours of closure (DZ), the three normal stresses (SIGZZ, SIGXX, SIGYY) and the strength/stress ratio (S.S.R.) are perhaps most useful at this stage.

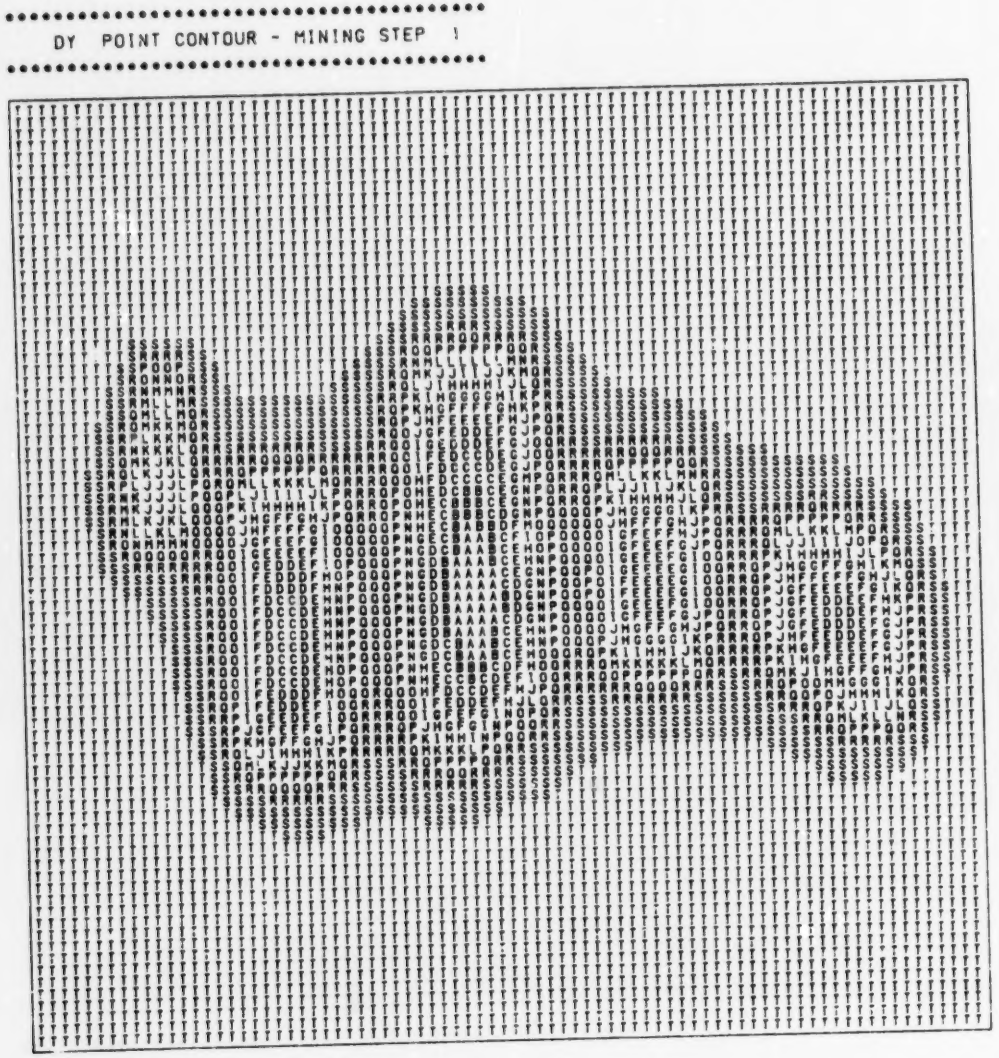
DX POINT CONTOUR - MINING STEP 1



SYMBOL	VALUE
A	-2.250E-04 - -2.000E-04
B	-2.000E-04 - -1.750E-04
C	-1.750E-04 - -1.500E-04
D	-1.500E-04 - -1.250E-04
E	-1.250E-04 - -1.000E-04
F	-1.000E-04 - -7.500E-05
G	-7.500E-05 - -5.000E-05
H	-5.000E-05 - -2.500E-05
I	-2.500E-05 - 0.
J	0. - 2.500E-05
K	2.500E-05 - 5.000E-05
L	5.000E-05 - 7.500E-05
M	7.500E-05 - 1.000E-04
N	1.000E-04 - 1.250E-04
O	1.250E-04 - 1.500E-04
P	1.500E-04 - 1.750E-04
Q	1.750E-04 - 2.000E-04
R	2.000E-04 - 2.250E-04

MINIMUM VALUE = -2.093E-04 M
MAXIMUM VALUE = 2.088E-04 M
CONTOUR INTERVAL = 2.500E-05 M

Fig. 5 - Plots of DX point contour - C.C.S. mine model, mining step 1



SYMBOL	VALUE
A	-6.000E-03 - -5.700E-03
B	-5.700E-03 - -5.400E-03
C	-5.400E-03 - -5.100E-03
D	-5.100E-03 - -4.800E-03
E	-4.800E-03 - -4.500E-03
F	-4.500E-03 - -4.200E-03
G	-4.200E-03 - -3.900E-03
H	-3.900E-03 - -3.600E-03
I	-3.600E-03 - -3.300E-03
J	-3.300E-03 - -3.000E-03
K	-3.000E-03 - -2.700E-03
L	-2.700E-03 - -2.400E-03
M	-2.400E-03 - -2.100E-03
N	-2.100E-03 - -1.800E-03
O	-1.800E-03 - -1.500E-03
P	-1.500E-03 - -1.200E-03
Q	-1.200E-03 - -9.000E-04
R	-9.000E-04 - -6.000E-04
S	-6.000E-04 - -3.000E-04
T	-3.000E-04 - 0.

MINIMUM VALUE = -5.971E-03 M
MAXIMUM VALUE = -6.394E-06 M
CONTOUR INTERVAL = 3.000E-04 M

Fig. 6 - Plots of DY point contour - C.C.S. mine model, mining step 1

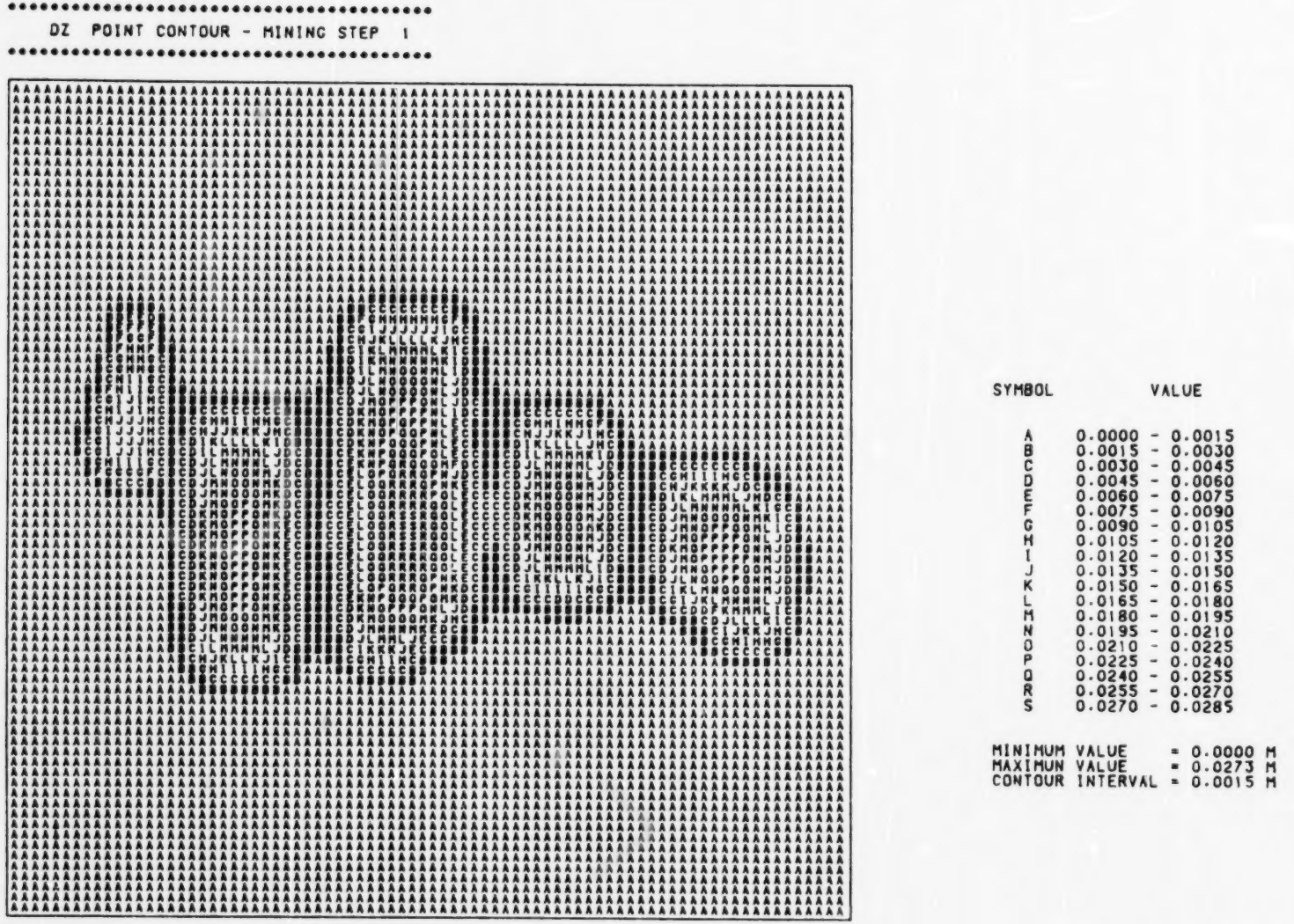
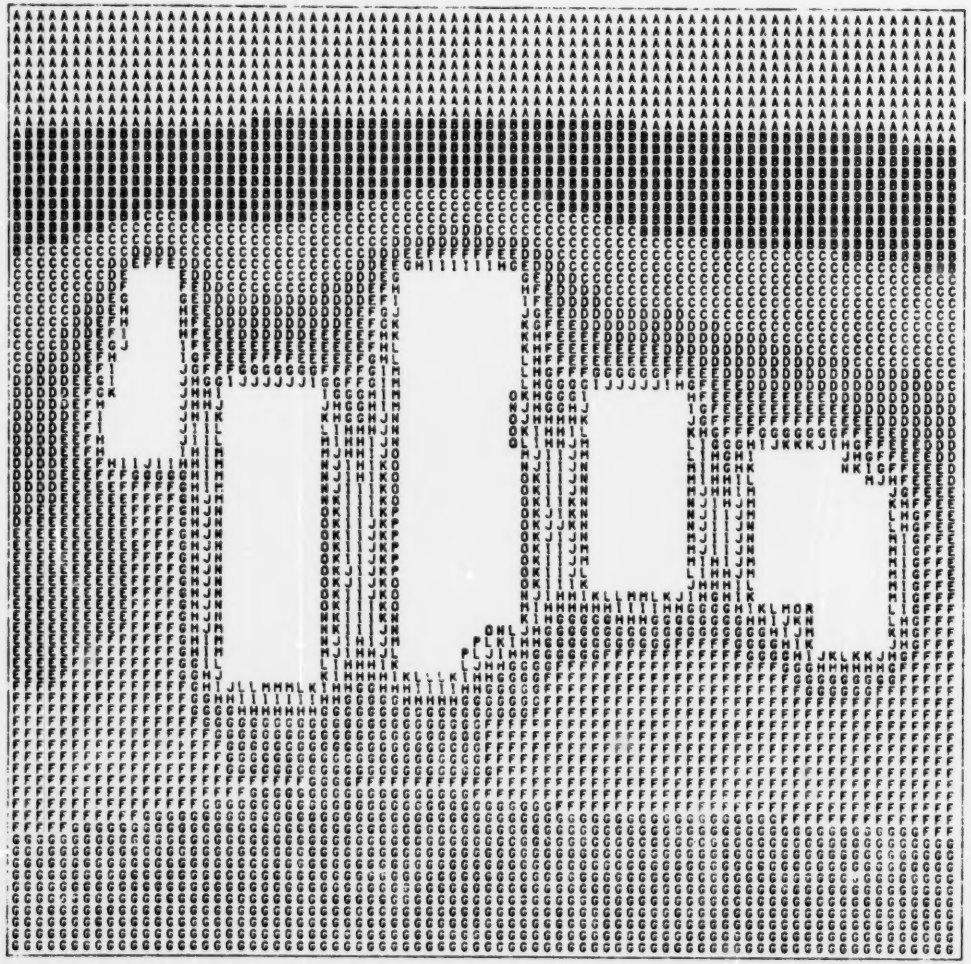


Fig. 7 - Plots of DZ point contour - C.C.S. mine model, mining step 1

SIGZZ POINT CONTOUR - MINING STEP 1

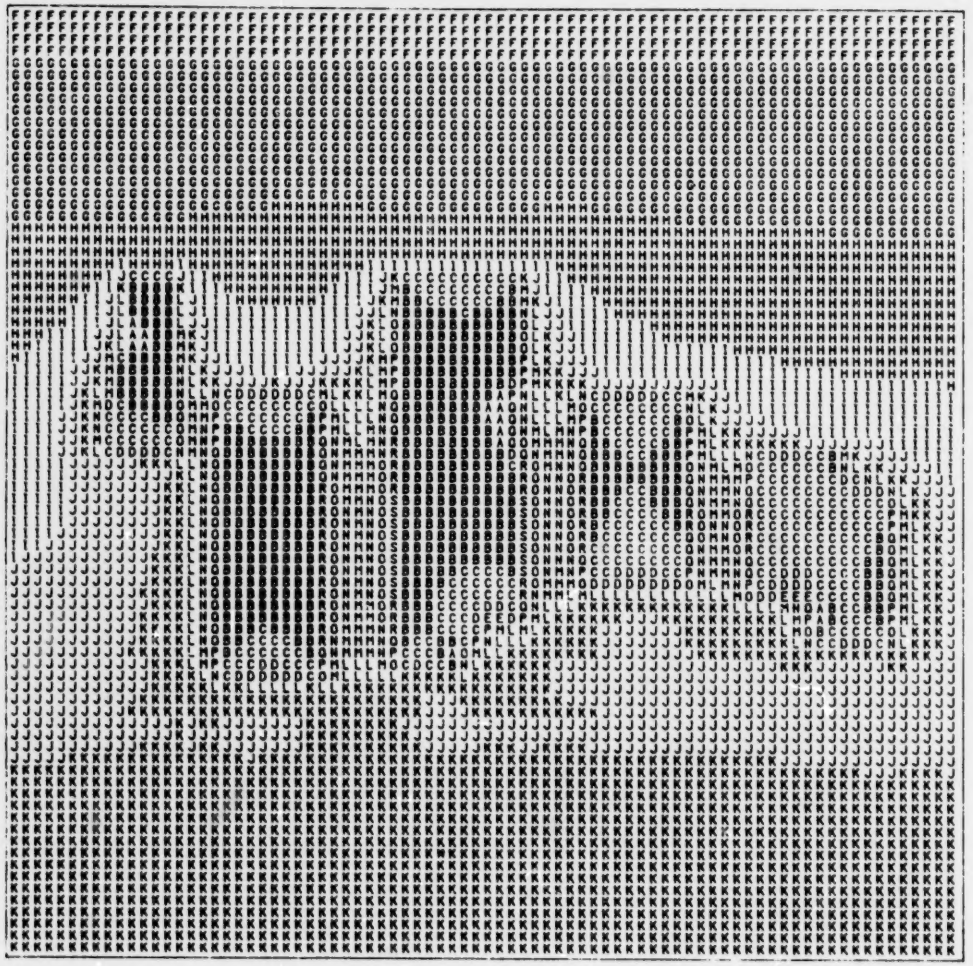


SYMBOL	VALUE
A	12.0000 - 14.0000
B	14.0000 - 16.0000
C	16.0000 - 18.0000
D	18.0000 - 20.0000
E	20.0000 - 22.0000
F	22.0000 - 24.0000
G	24.0000 - 26.0000
H	26.0000 - 28.0000
I	28.0000 - 30.0000
J	30.0000 - 32.0000
K	32.0000 - 34.0000
L	34.0000 - 36.0000
M	36.0000 - 38.0000
N	38.0000 - 40.0000
O	40.0000 - 42.0000
P	42.0000 - 44.0000
Q	44.0000 - 46.0000
R	46.0000 - 48.0000

MINIMUM VALUE = 12.2064 MPA
MAXIMUN VALUE = 46.75?? MPA
CONTOUR INTERVAL = 2.0000 MPA

Fig. 8 - Plots of SIGZZ point contour - C.C.S. mine model, mining step 1

SIGXX POINT CONTOUR - MINING STEP 1

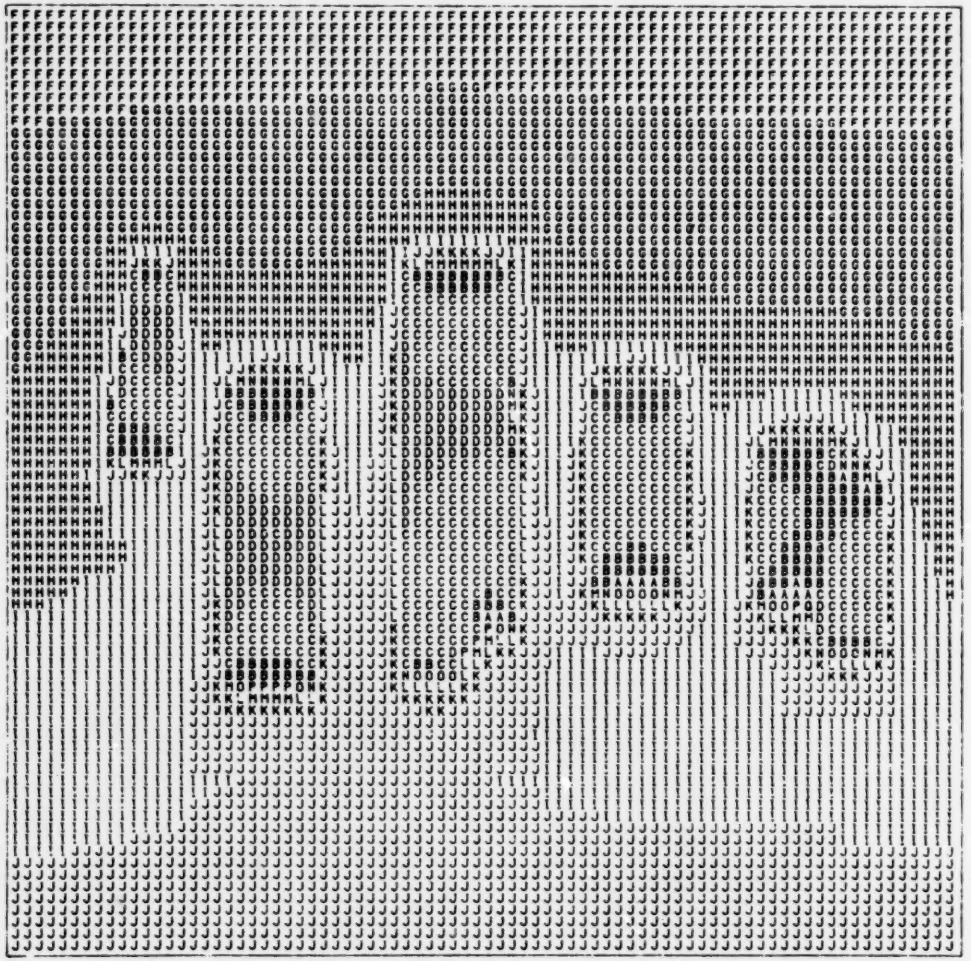


SYMBOL	VALUE
A	-2.5000 - 0.0000
B	0.0000 - 2.5000
C	2.5000 - 5.0000
D	5.0000 - 7.5000
E	7.5000 - 10.0000
F	10.0000 - 12.5000
G	12.5000 - 15.0000
H	15.0000 - 17.5000
I	17.5000 - 20.0000
J	20.0000 - 22.5000
K	22.5000 - 25.0000
L	25.0000 - 27.5000
M	27.5000 - 30.0000
N	30.0000 - 32.5000
O	32.5000 - 35.0000
P	35.0000 - 37.5000
Q	37.5000 - 40.0000
R	40.0000 - 42.5000
S	42.5000 - 45.0000

MINIMUM VALUE = -1.0560 MPA
MAXIMUN VALUE = 43.3058 MPA
CONTOUR INTERVAL = 2.5000 MPA

Fig. 9 - Plots of SIGXX point contour - C.C.S. mine model, mining step 1

SIGYY POINT CONTOUR - MINING STEP 1



SYMBOL	VALUE
A	-5.0000 - -2.5000
B	-2.5000 - 0.0000
C	0.0000 - 2.5000
D	2.5000 - 5.0000
E	5.0000 - 7.5000
F	7.5000 - 10.0000
G	10.0000 - 12.5000
H	12.5000 - 15.0000
I	15.0000 - 17.5000
J	17.5000 - 20.0000
K	20.0000 - 22.5000
L	22.5000 - 25.0000
M	25.0000 - 27.5000
N	27.5000 - 30.0000
O	30.0000 - 32.5000
P	32.5000 - 35.0000
Q	35.0000 - 37.5000

MINIMUM VALUE = -4.7213 MPA
MAXIMUN VALUE = 35.1064 MPA
CONTOUR INTERVAL = 2.5000 MPA

Fig. 10 - Plots of SIGYY point contour - C.C.S. mine model, mining step 1

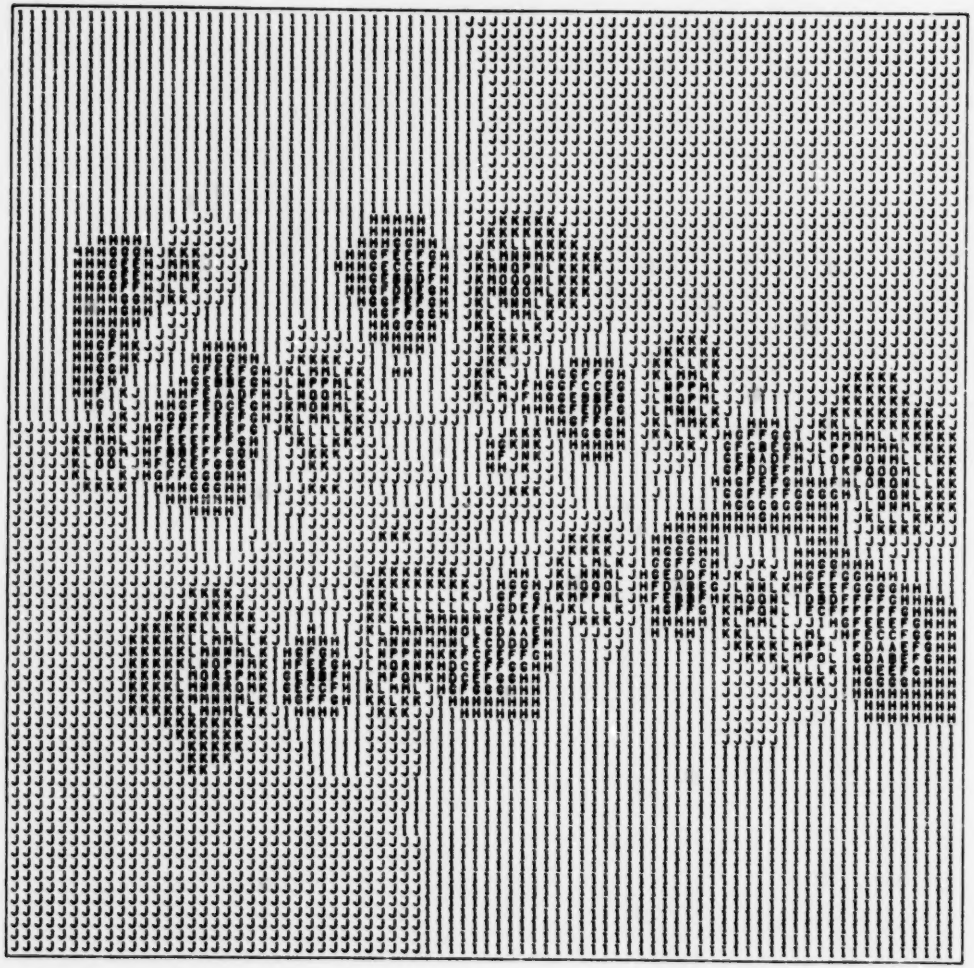
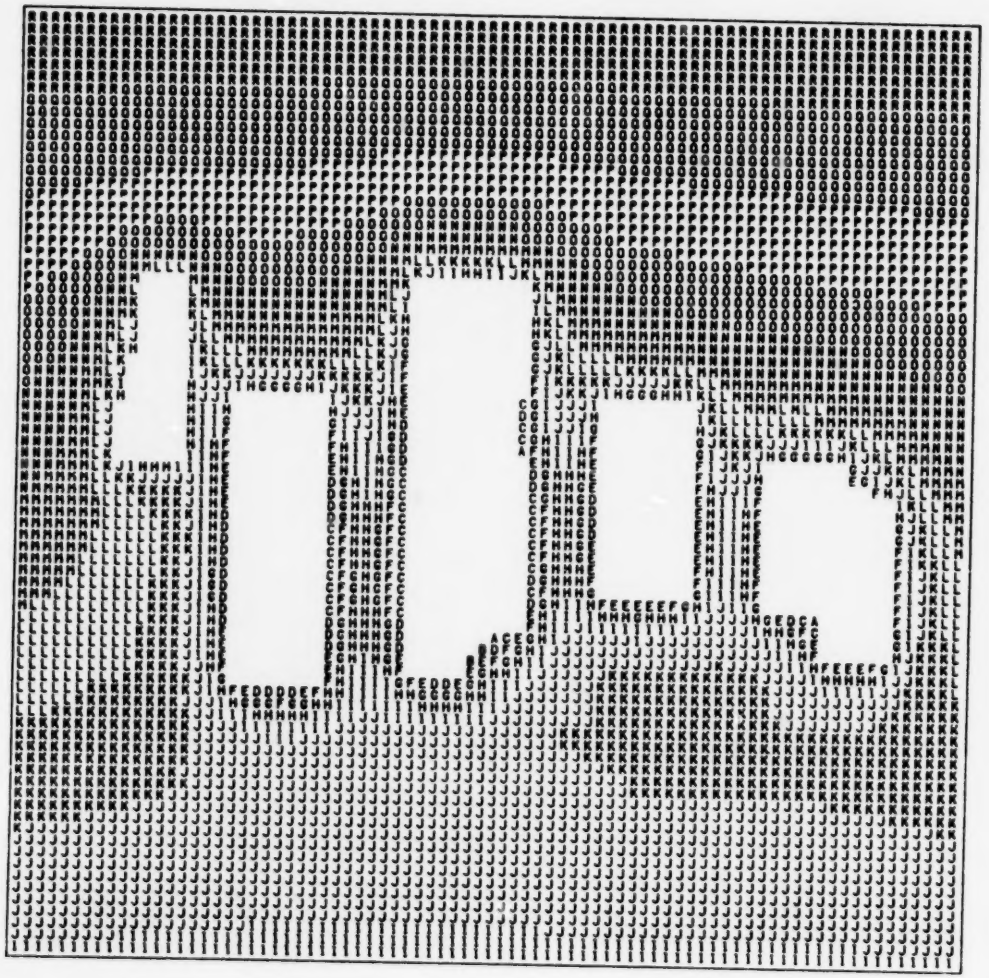


Fig. 11 - Plots of SIGXY point contour - C.C.S. mine model, mining step 1

SIGYZ POINT CONTOUR - MINING STEP 1

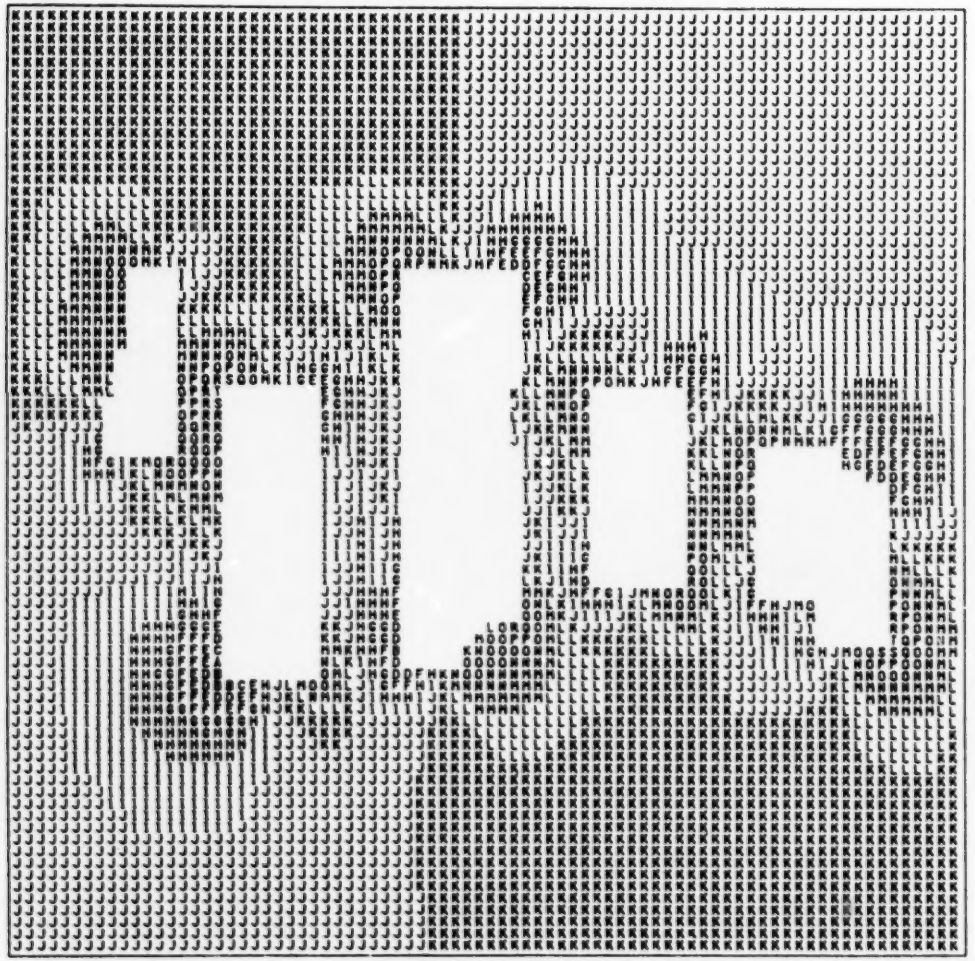


SYMBOL	VALUE
A	-6.5000 - -6.2500
B	-6.2500 - -6.0000
C	-6.0000 - -5.7500
D	-5.7500 - -5.5000
E	-5.5000 - -5.2500
F	-5.2500 - -5.0000
G	-5.0000 - -4.7500
H	-4.7500 - -4.5000
I	-4.5000 - -4.2500
J	-4.2500 - -4.0000
K	-4.0000 - -3.7500
L	-3.7500 - -3.5000
M	-3.5000 - -3.2500
N	-3.2500 - -3.0000
O	-3.0000 - -2.7500
P	-2.7500 - -2.5000
Q	-2.5000 - -2.2500
R	-2.2500 - -2.0000

MINIMUM VALUE = -6.3671 MPA
MAXIMUM VALUE = -2.0405 MPA
CONTOUR INTERVAL = 0.2500 MPA

Fig. 12 - Plots of SIGYZ point contour - C.C.S. mine model, mining step 1

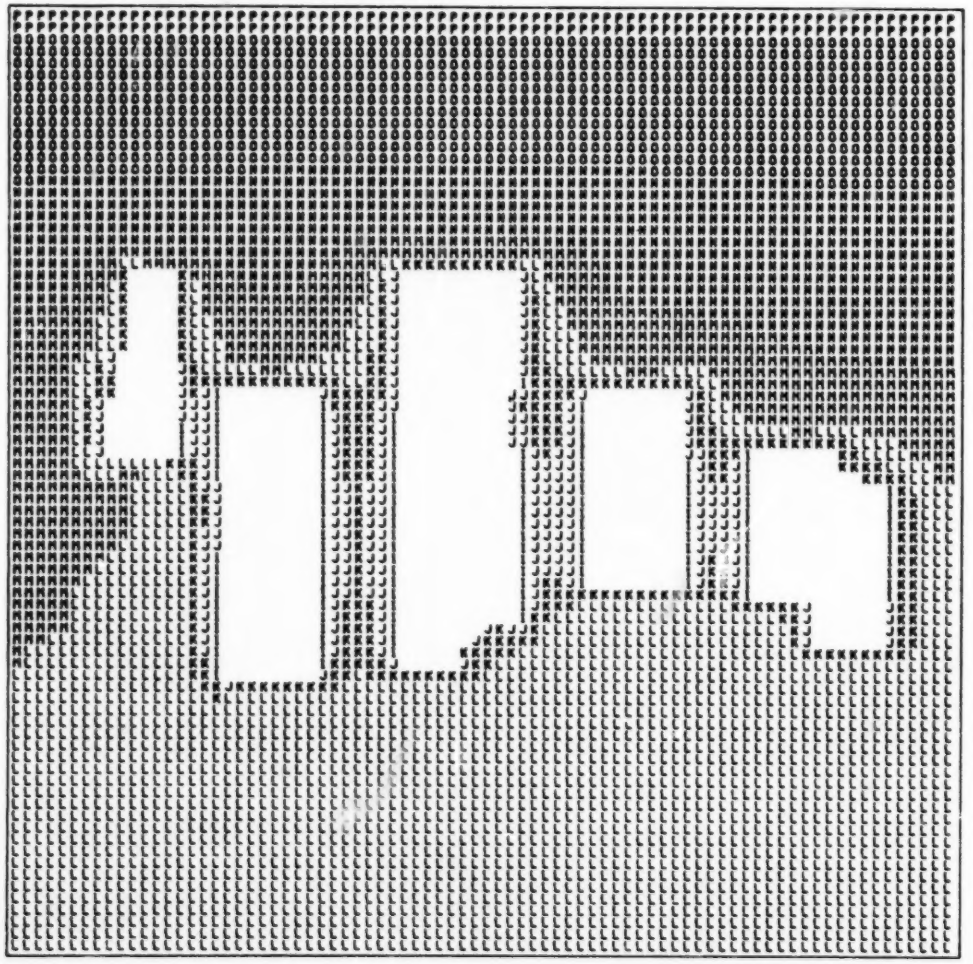
SIGZX POINT CONTOUR - MINING STEP 1



SYMBOL	VALUE
A	-0.2500 - -0.2250
B	-0.2250 - -0.2000
C	-0.2000 - -0.1750
D	-0.1750 - -0.1500
E	-0.1500 - -0.1250
F	-0.1250 - -0.1000
G	-0.1000 - -0.0750
H	-0.0750 - -0.0500
I	-0.0500 - -0.0250
J	-0.0250 - 0.0000
K	0.0000 - 0.0250
L	0.0250 - 0.0500
M	0.0500 - 0.0750
N	0.0750 - 0.1000
O	0.1000 - 0.1250
P	0.1250 - 0.1500
Q	0.1500 - 0.1750
R	0.1750 - 0.2000
S	0.2000 - 0.2250
T	0.2250 - 0.2500

MINIMUM VALUE = -0.2278 MPA
MAXIMUN VALUE = 0.2322 MPA
CONTOUR INTERVAL = 0.0250 MPA

Fig. 13 - Plots of SIGZX point contour - C.C.S. mine model, mining step 1

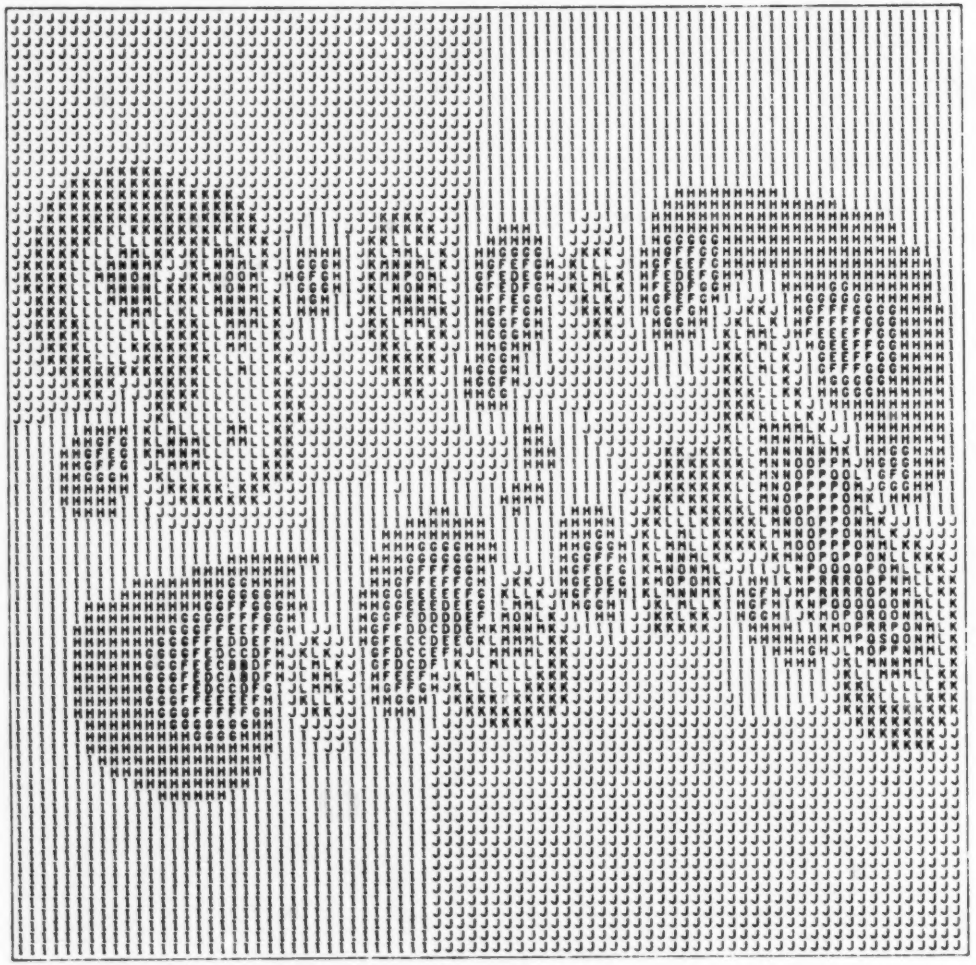


SYMBOL	VALUE		
A	0.	-	.2000E+00
B	.2000E+00	-	.4000E+00
C	.4000E+00	-	.6000E+00
D	.6000E+00	-	.8000E+00
E	.8000E+00	-	.1000E+01
F	.1000E+01	-	.1200E+01
G	.1200E+01	-	.1400E+01
H	.1400E+01	-	.1600E+01
I	.1600E+01	-	.1800E+01
J	.1800E+01	-	.2000E+01
K	.2000E+01	-	.2200E+01
L	.2200E+01	-	.2400E+01
M	.2400E+01	-	.2600E+01
N	.2600E+01	-	.2800E+01
O	.2800E+01	-	.3000E+01
P	.3000E+01	-	.3200E+01
Q	.3200E+01	-	.3400E+01

MINIMUM VALUE	= 0.0000
MAXIMUM VALUE	= 3.2626
CONTOUR INTERVAL	= 0.2000
COMP. STRENGTH	= 75.0000 MPA
CONSTANT M	= 1.7000
CONSTANT S	= 0.0040

Fig. 14 - Plots of S.S.R. point contour - C.C.S. mine model, mining step 1

DX POINT CONTOUR - MINING STEP 2

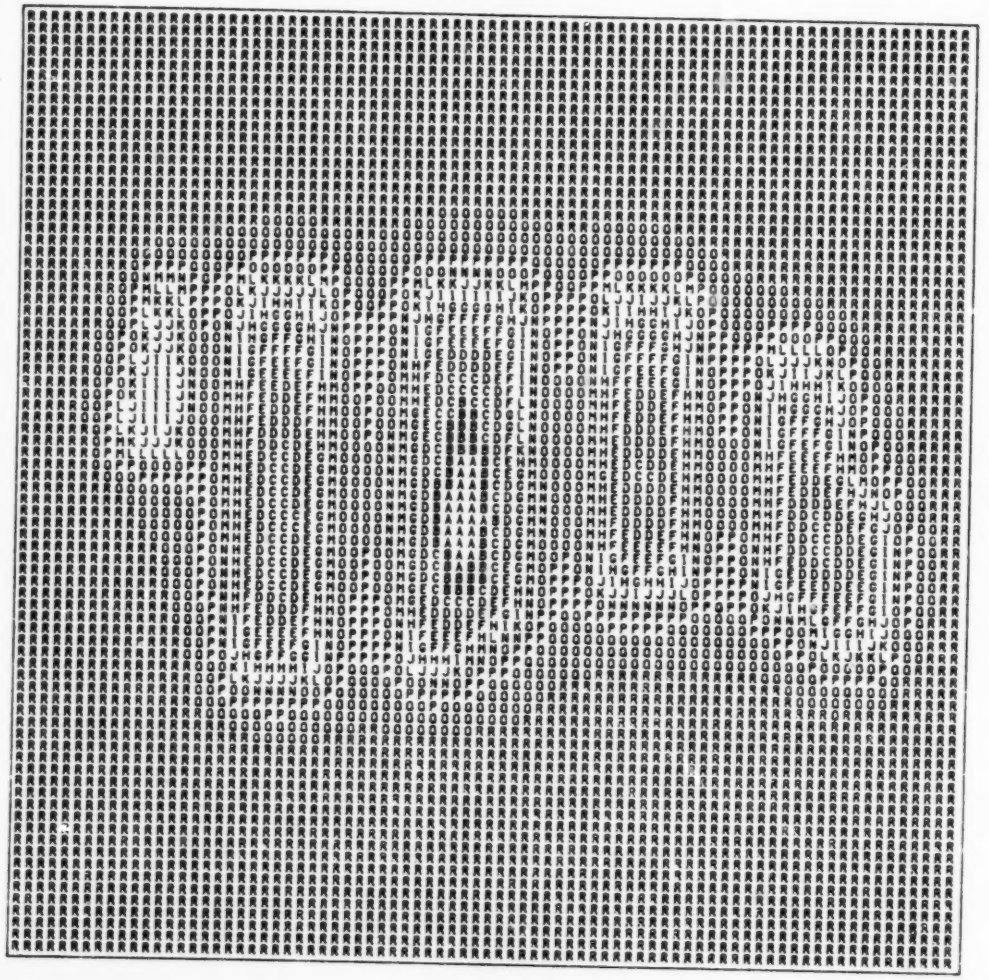


SYMBOL	VALUE
A	-2.250E-04 - -2.000E-04
B	-2.000E-04 - -1.750E-04
C	-1.750E-04 - -1.500E-04
D	-1.500E-04 - -1.250E-04
E	-1.250E-04 - -1.000E-04
F	-1.000E-04 - -7.500E-05
G	-7.500E-05 - -5.000E-05
H	-5.000E-05 - -2.500E-05
I	-2.500E-05 - 0.
J	0. - 2.500E-05
K	2.500E-05 - 5.000E-05
L	5.000E-05 - 7.500E-05
M	7.500E-05 - 1.000E-04
N	1.000E-04 - 1.250E-04
O	1.250E-04 - 1.500E-04
P	1.500E-04 - 1.750E-04
Q	1.750E-04 - 2.000E-04
R	2.000E-04 - 2.250E-04
S	2.250E-04 - 2.500E-04

MINIMUM VALUE = -2.171E-04 M
MAXIMUM VALUE = 2.286E-04 M
CONTOUR INTERVAL = 2.500E-05 M

Fig. 15 - Plots of DX point contour - C.C.S. mine model, mining step 2

DY POINT CONTOUR - MINING STEP 2

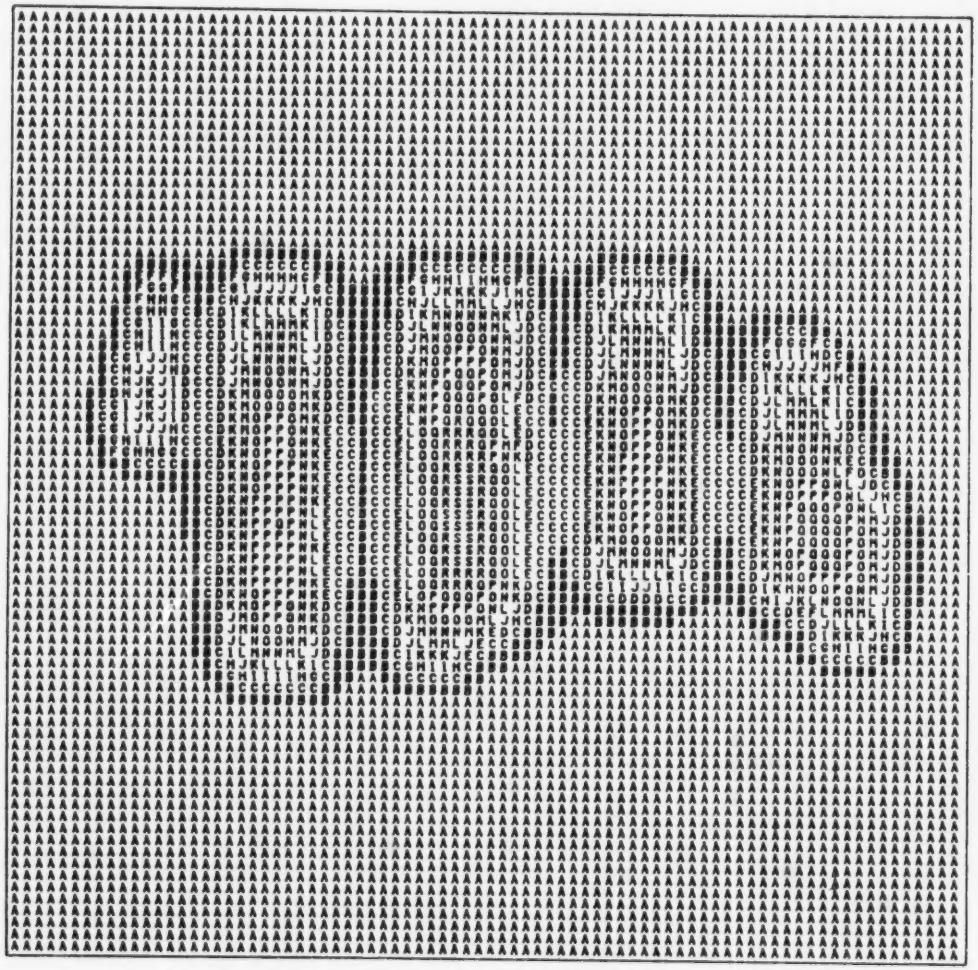


SYMBOL	VALUE
A	-6.300E-03 - -5.950E-03
B	-5.950E-03 - -5.600E-03
C	-5.600E-03 - -5.250E-03
D	-5.250E-03 - -4.900E-03
E	-4.900E-03 - -4.550E-03
F	-4.550E-03 - -4.200E-03
G	-4.200E-03 - -3.850E-03
H	-3.850E-03 - -3.500E-03
I	-3.500E-03 - -3.150E-03
J	-3.150E-03 - -2.800E-03
K	-2.800E-03 - -2.450E-03
L	-2.450E-03 - -2.100E-03
M	-2.100E-03 - -1.750E-03
N	-1.750E-03 - -1.400E-03
O	-1.400E-03 - -1.050E-03
P	-1.050E-03 - -7.000E-04
Q	-7.000E-04 - -3.500E-04
R	-3.500E-04 - 0.

MINIMUM VALUE = -6.153E-03 M
MAXIMUM VALUE = -9.491E-06 M
CONTOUR INTERVAL = 3.500E-04 M

Fig. 16 - Plots of DY point contour - C.C.S. mine model, mining step 2

DZ POINT CONTOUR - MINING STEP 2

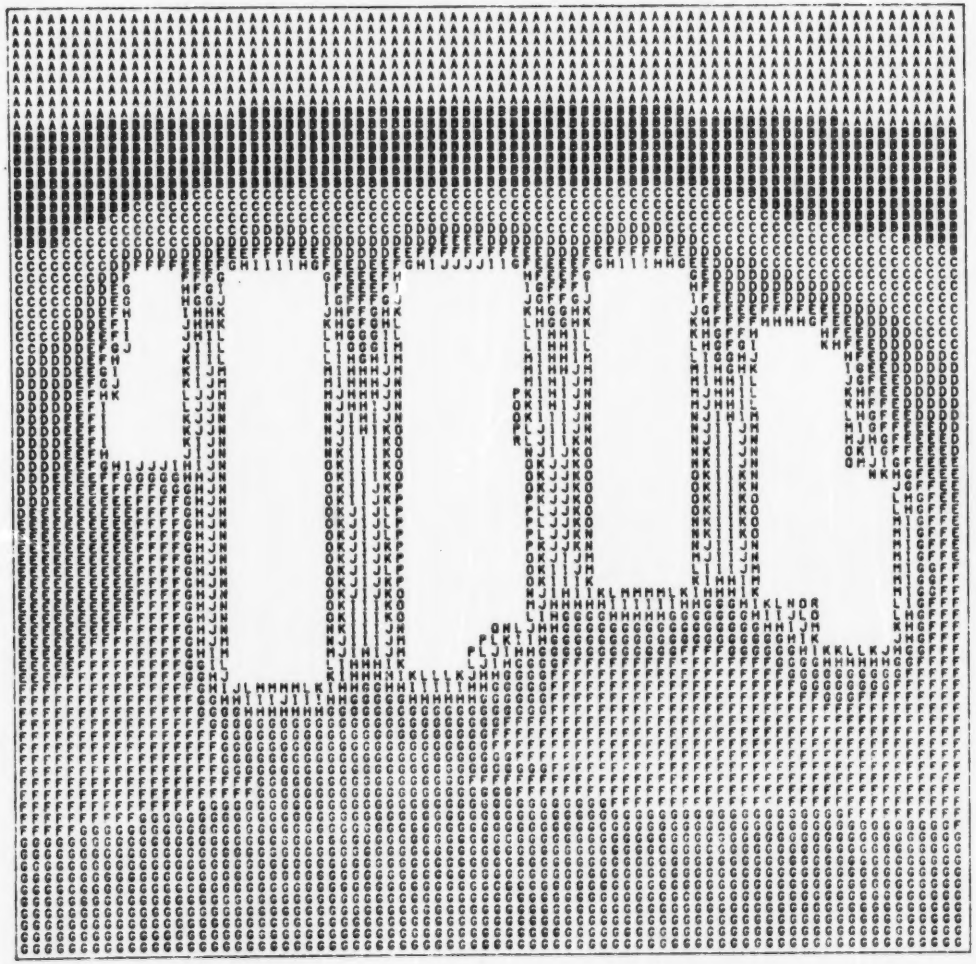


SYMBOL	VALUE
A	0.0000 - 0.0015
B	0.0015 - 0.0030
C	0.0030 - 0.0045
D	0.0045 - 0.0060
E	0.0060 - 0.0075
F	0.0075 - 0.0090
G	0.0090 - 0.0105
H	0.0105 - 0.0120
I	0.0120 - 0.0135
J	0.0135 - 0.0150
K	0.0150 - 0.0165
L	0.0165 - 0.0180
M	0.0180 - 0.0195
N	0.0195 - 0.0210
O	0.0210 - 0.0225
P	0.0225 - 0.0240
Q	0.0240 - 0.0255
R	0.0255 - 0.0270
S	0.0270 - 0.0285

MINIMUM VALUE = 0.0000 M
MAXIMUN VALUE = 0.0279 M
CONTOUR INTERVAL = 0.0015 M

Fig. 17 - Plots of DZ point contour - C.C.S. mine model, mining step 2

SIGZZ POINT CONTOUR - MINING STEP 2

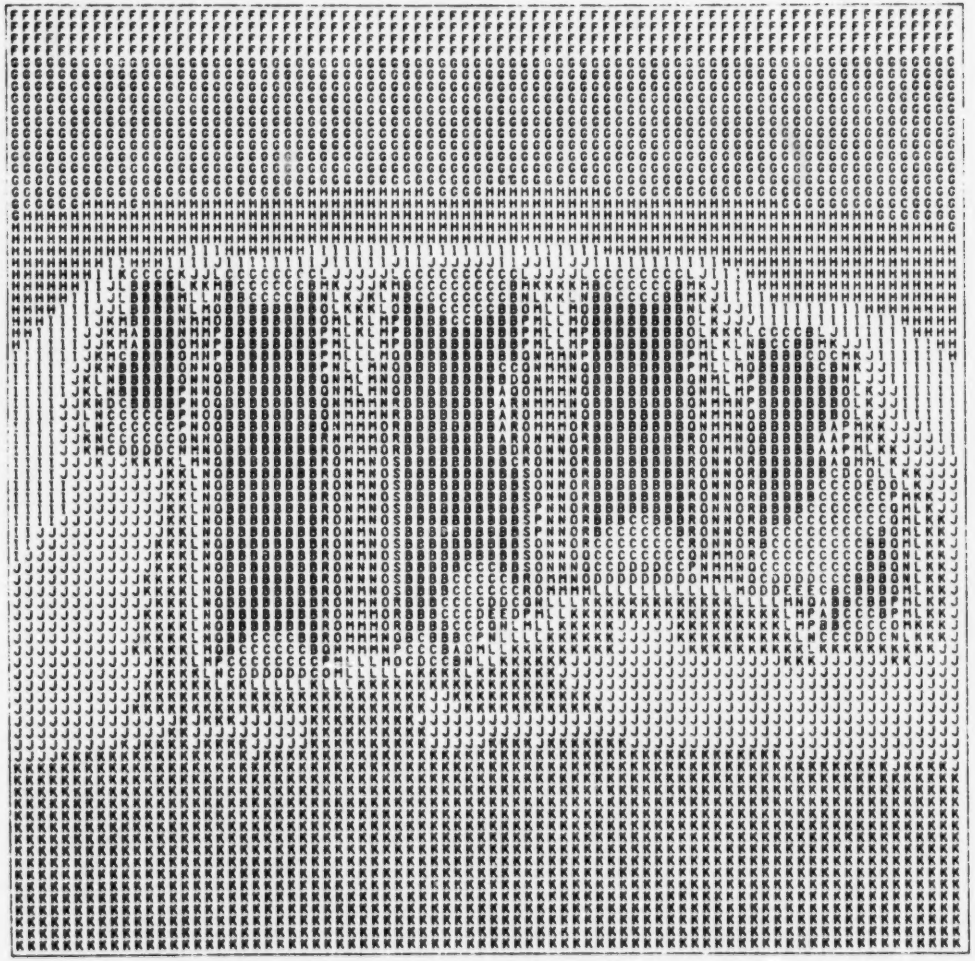


SYMBOL	VALUE
A	12.0000 - 14.0000
B	14.0000 - 16.0000
C	16.0000 - 18.0000
D	18.0000 - 20.0000
E	20.0000 - 22.0000
F	22.0000 - 24.0000
G	24.0000 - 26.0000
H	26.0000 - 28.0000
I	28.0000 - 30.0000
J	30.0000 - 32.0000
K	32.0000 - 34.0000
L	34.0000 - 36.0000
M	36.0000 - 38.0000
N	38.0000 - 40.0000
O	40.0000 - 42.0000
P	42.0000 - 44.0000
Q	44.0000 - 46.0000
R	46.0000 - 48.0000

MINIMUM VALUE = 12.2396 MPA
MAXIMUN VALUE = 47.7029 MPA
CONTOUR INTERVAL = 2.0000 MPA

Fig. 18 - Plots of SIGZZ point contour - C.C.S. mine model, mining step 2

SIGXX POINT CONTOUR - MINING STEP 2

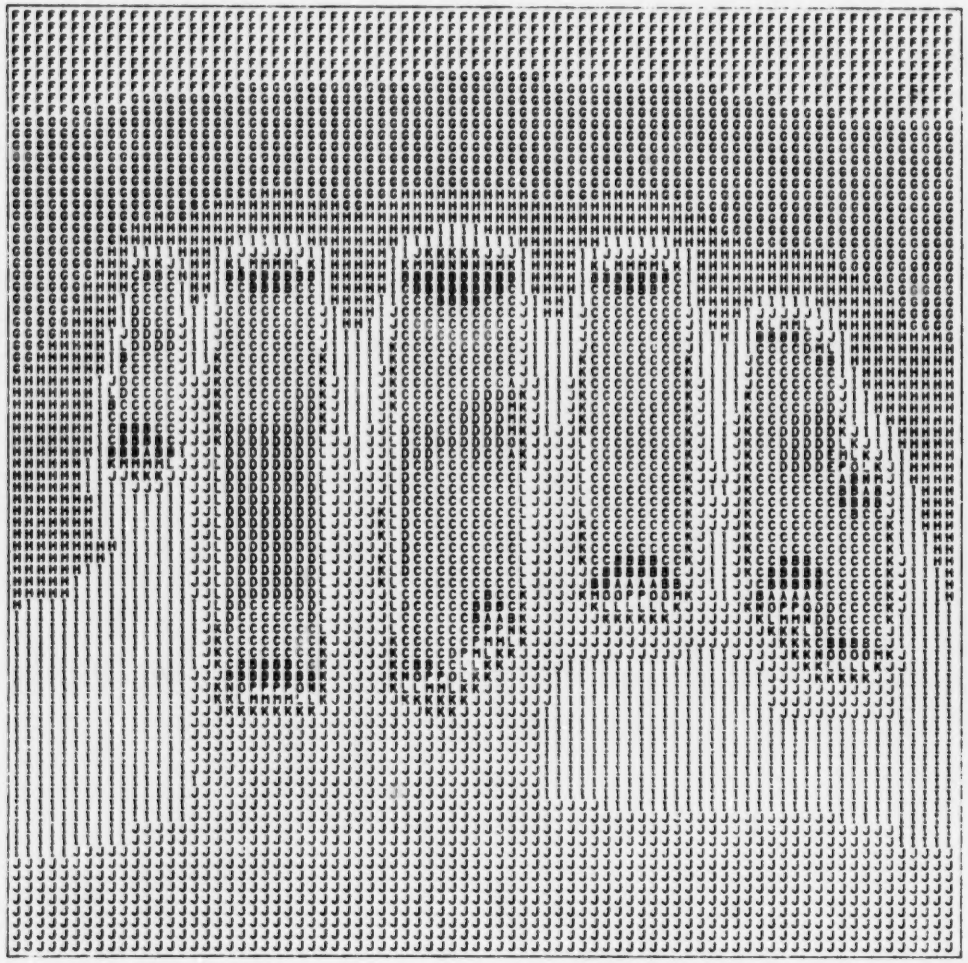


SYMBOL	VALUE
A	-2.5000 - 0.0000
B	0.0000 - 2.5000
C	2.5000 - 5.0000
D	5.0000 - 7.5000
E	7.5000 - 10.0000
F	10.0000 - 12.5000
G	12.5000 - 15.0000
H	15.0000 - 17.5000
I	17.5000 - 20.0000
J	20.0000 - 22.5000
K	22.5000 - 25.0000
L	25.0000 - 27.5000
M	27.5000 - 30.0000
N	30.0000 - 32.5000
O	32.5000 - 35.0000
P	35.0000 - 37.5000
Q	37.5000 - 40.0000
R	40.0000 - 42.5000
S	42.5000 - 45.0000

MINIMUM VALUE = -1.8194 MPA
MAXIMUM VALUE = 44.1198 MPA
CONTOUR INTERVAL = 2.5000 MPA

Fig. 19 - Plots of SIGXX point contour - C.C.S. mine model, mining step 2

SIGYY POINT CONTOUR - MINING STEP 2

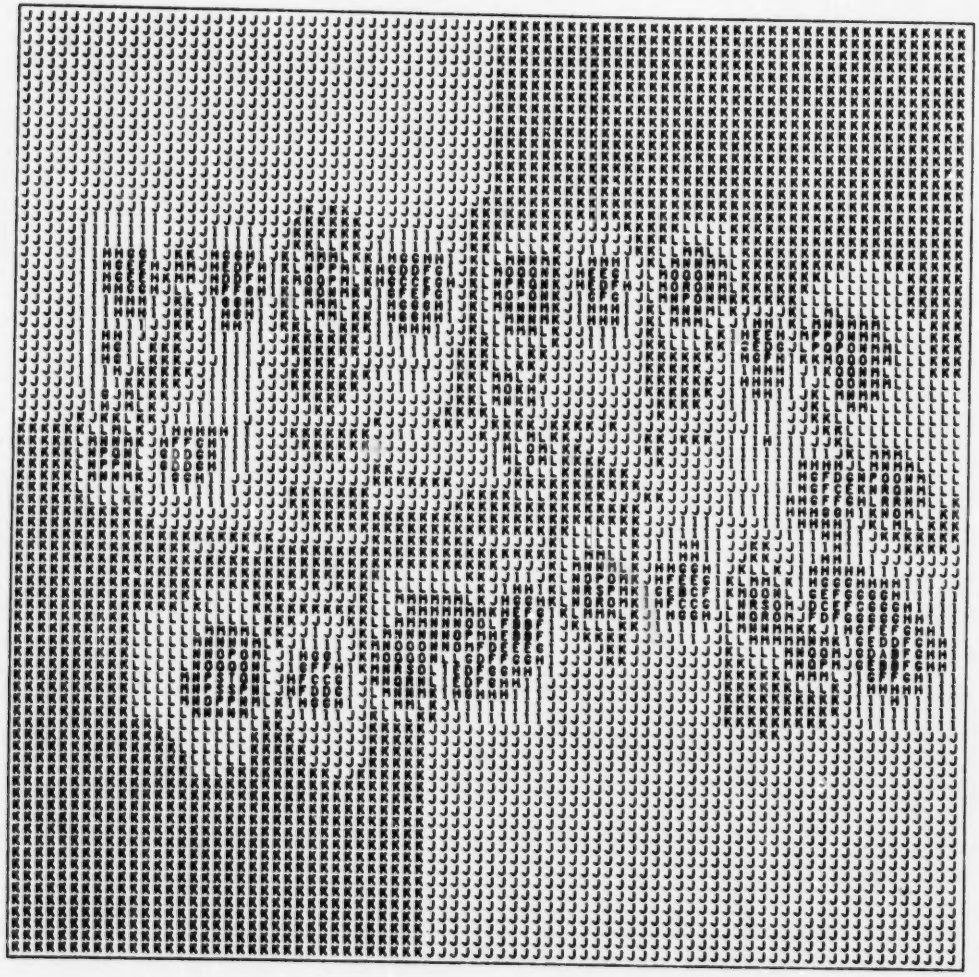


SYMBOL	VALUE
A	-5.0000 - -2.5000
B	-2.5000 - 0.0000
C	0.0000 - 2.5000
D	2.5000 - 5.0000
E	5.0000 - 7.5000
F	7.5000 - 10.0000
G	10.0000 - 12.5000
H	12.5000 - 15.0000
I	15.0000 - 17.5000
J	17.5000 - 20.0000
K	20.0000 - 22.5000
L	22.5000 - 25.0000
M	25.0000 - 27.5000
N	27.5000 - 30.0000
O	30.0000 - 32.5000
P	32.5000 - 35.0000
Q	35.0000 - 37.5000

MINIMUM VALUE = -4.5159 MPA
MAXIMUN VALUE = 36.0678 MPA
CONTOUR INTERVAL = 2.5000 MPA

Fig. 20 - Plots of SIGYY point contour - C.C.S. mine model, mining step 2

SIGXY POINT CONTOUR - MINING STEP 2

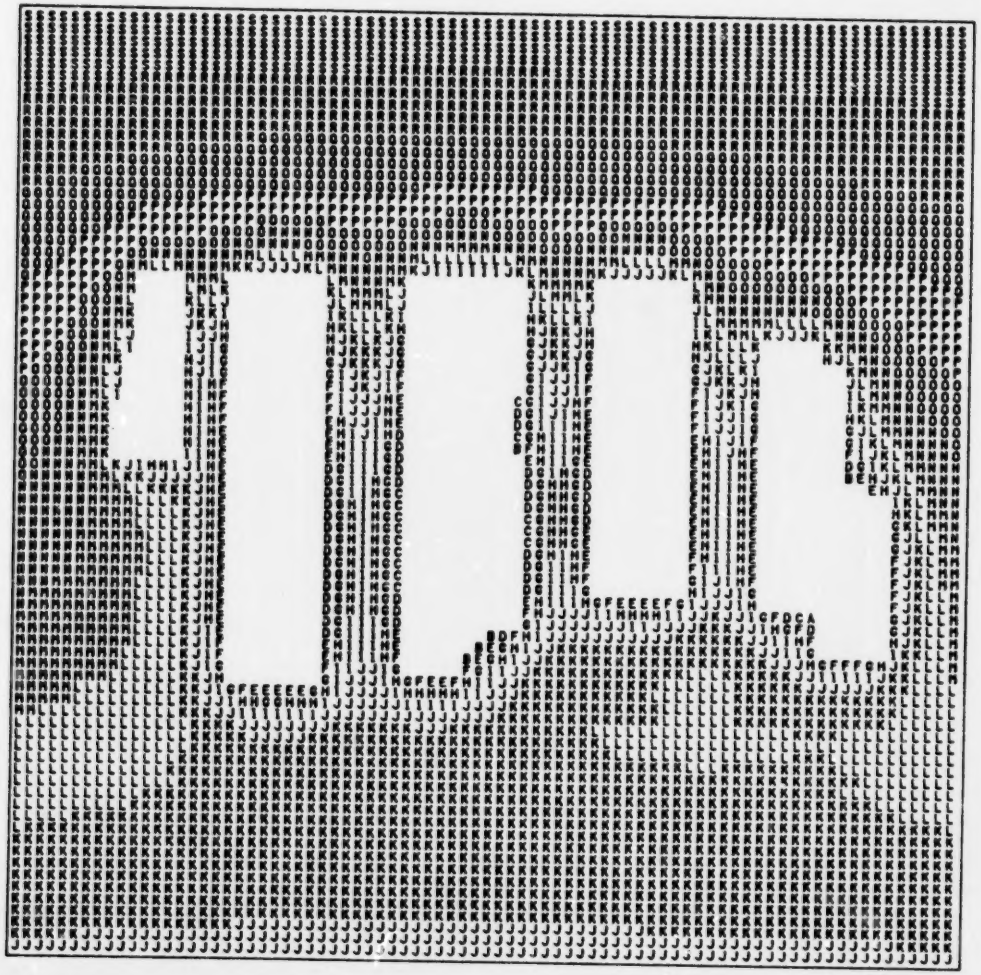


SYMBOL	VALUE
A	-5.5000 - -4.9500
B	-4.9500 - -4.4000
C	-4.4000 - -3.8500
D	-3.8500 - -3.3000
E	-3.3000 - -2.7500
F	-2.7500 - -2.2000
G	-2.2000 - -1.6500
H	-1.6500 - -1.1000
I	-1.1000 - -0.5500
J	-0.5500 - 0.0000
K	0.0000 - 0.5500
L	0.5500 - 1.1000
M	1.1000 - 1.6500
N	1.6500 - 2.2000
O	2.2000 - 2.7500
P	2.7500 - 3.3000
Q	3.3000 - 3.8500
R	3.8500 - 4.4000
S	4.4000 - 4.9500
T	4.9500 - 5.5000

MINIMUM VALUE = -5.1337 MPA
MAXIMUN VALUE = 5.1050 MPA
CONTOUR INTERVAL = 0.5500 MPA

Fig. 21 - Plots of SIGXY point contour - C.C.S. mine model, mining step 2

SIGYZ POINT CONTOUR - MINING STEP 2

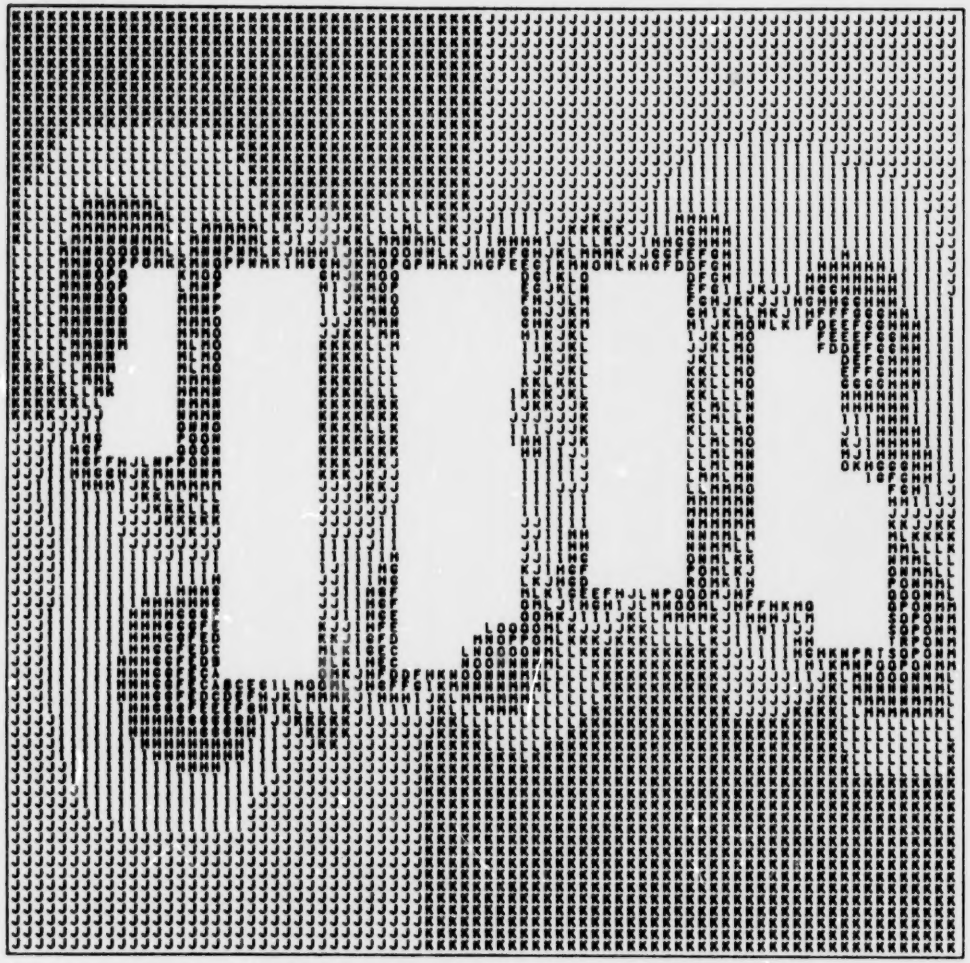


SYMBOL	VALUE
A	-6.7500 - -6.5000
B	-6.5000 - -6.2500
C	-6.2500 - -6.0000
D	-6.0000 - -5.7500
E	-5.7500 - -5.5000
F	-5.5000 - -5.2500
G	-5.2500 - -5.0000
H	-5.0000 - -4.7500
I	-4.7500 - -4.5000
J	-4.5000 - -4.2500
K	-4.2500 - -4.0000
L	-4.0000 - -3.7500
M	-3.7500 - -3.5000
N	-3.5000 - -3.2500
O	-3.2500 - -3.0000
P	-3.0000 - -2.7500
Q	-2.7500 - -2.5000
R	-2.5000 - -2.2500
S	-2.2500 - -2.0000

MINIMUM VALUE = -6.5754 MPA
MAXIMUN VALUE = -2.0469 MPA
CONTOUR INTERVAL = 0.2500 MPA

Fig. 22 - Plots of SIGYZ point contour - C.C.S. mine model, mining step 2

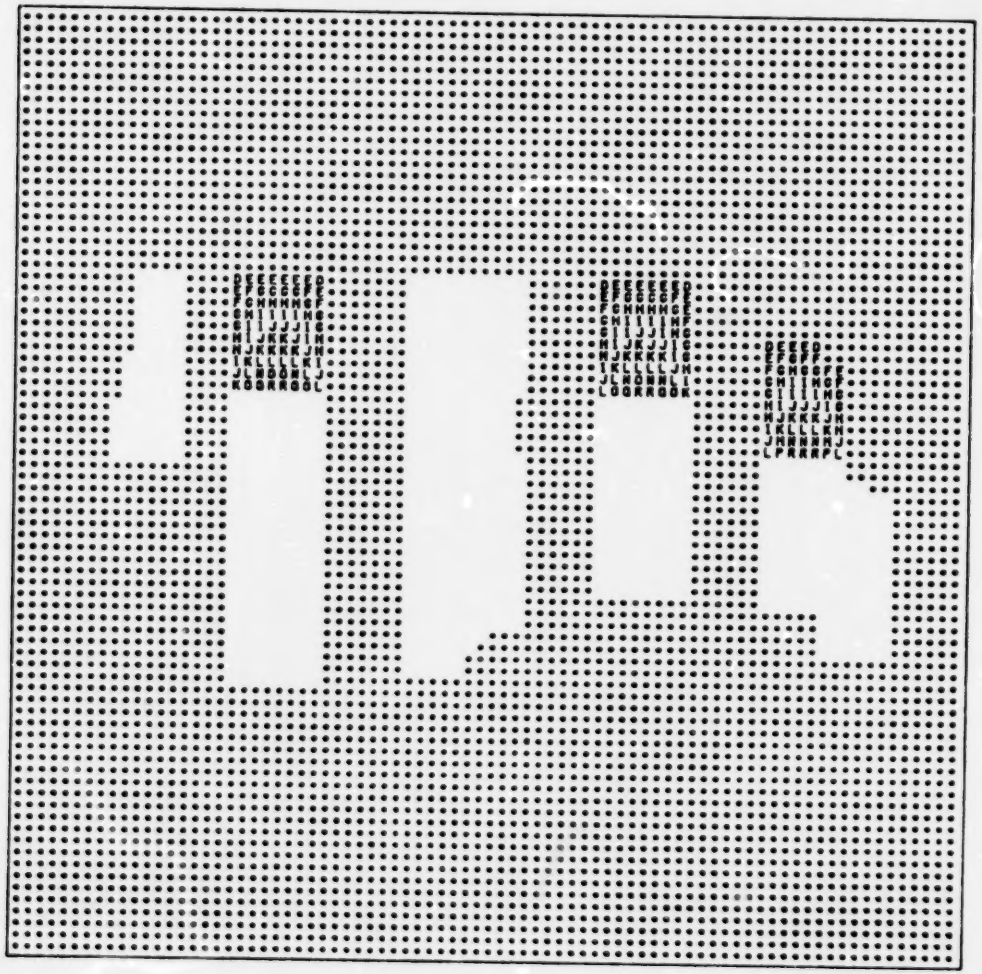
SIGZX POINT CONTOUR - MINING STEP 2



SYMBOL	VALUE
A	-0.2500 - -0.2250
B	-0.2250 - -0.2000
C	-0.2000 - -0.1750
D	-0.1750 - -0.1500
E	-0.1500 - -0.1250
F	-0.1250 - -0.1000
G	-0.1000 - -0.0750
H	-0.0750 - -0.0500
I	-0.0500 - -0.0250
J	-0.0250 - 0.0000
K	0.0000 - 0.0250
L	0.0250 - 0.0500
M	0.0500 - 0.0750
N	0.0750 - 0.1000
O	0.1000 - 0.1250
P	0.1250 - 0.1500
Q	0.1500 - 0.1750
R	0.1750 - 0.2000
S	0.2000 - 0.2250
T	0.2250 - 0.2500

MINIMUM VALUE = -0.2401 MPA
MAXIMUN VALUE = 0.2474 MPA
CONTOUR INTERVAL = 0.0250 MPA

Fig. 23 - Plots of SIGZX point contour - C.C.S. mine model, mining step 2



SYMBOL	VALUE
A	0.0000 - 0.0200
B	0.0200 - 0.0400
C	0.0400 - 0.0600
D	0.0600 - 0.0800
E	0.0800 - 0.1000
F	0.1000 - 0.1200
G	0.1200 - 0.1400
H	0.1400 - 0.1600
I	0.1600 - 0.1800
J	0.1800 - 0.2000
K	0.2000 - 0.2200
L	0.2200 - 0.2400
M	0.2400 - 0.2600
N	0.2600 - 0.2800
O	0.2800 - 0.3000
P	0.3000 - 0.3200
Q	0.3200 - 0.3400
R	0.3400 - 0.3600

MINIMUM VALUE = 0.0000 MJ/M•M
MAXIMUN VALUE = 0.3598 MJ/M•M
CONTOUR INTERVAL = 0.0200 MJ/M•M

Fig. 24 - Plots of ENERGY point contour - C.C.S. mine model, mining step 2

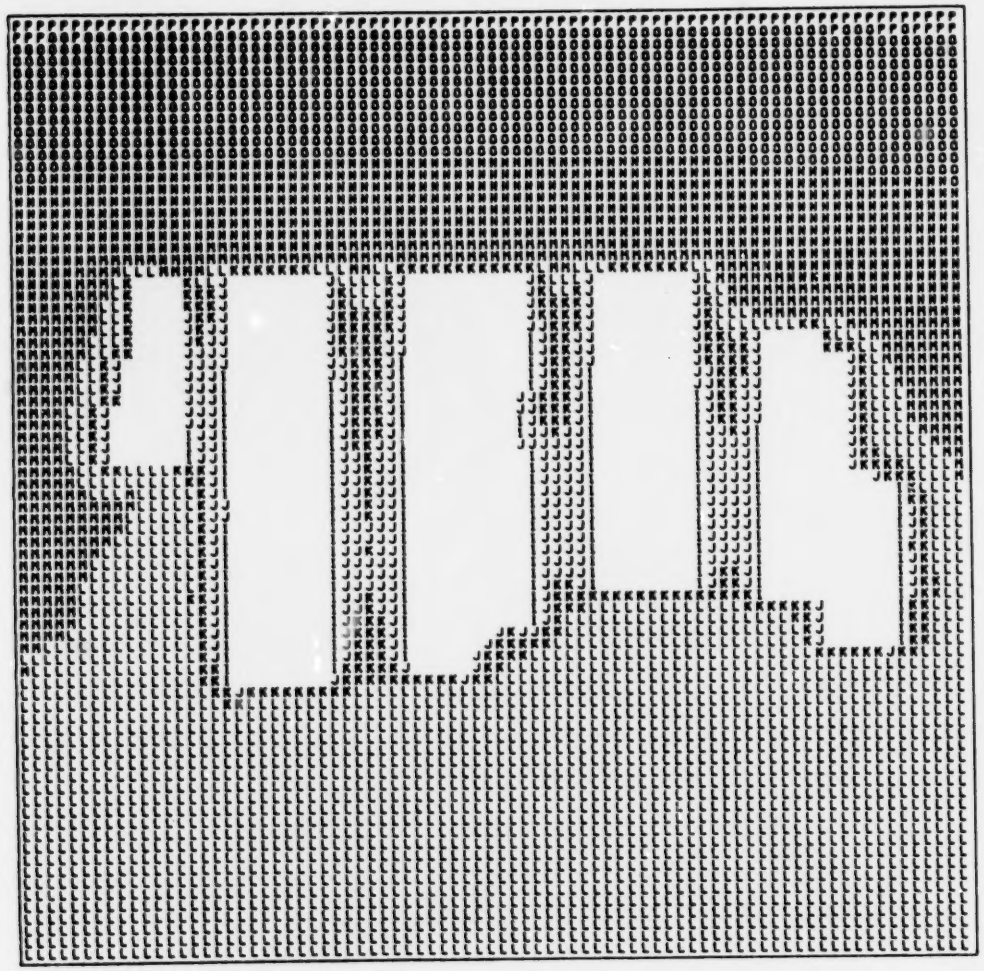


Fig. 25 - Plots of S.S.R. point contour - C.C.S. mine model, mining step 2

6.2 FAILURE CRITERIA

The design of a stope or opening in the mining horizon requires an assessment of the likely response of the rock mass to a set of induced stresses due to mining in adjacent areas. In order to predict this response a knowledge of the complete stress-strain behaviour and strength characteristics of the rock mass is required. A number of failure criteria have been developed for rocks but none of them is completely satisfactory for all types of rocks.

An empirical strength criteria for rock masses taking into account the rock qualities, has been proposed by Hoek et al (7), based on a wide range of experimental data. This approach is similar to the non-linear Mohr-Coulomb failure theory. Essentially the principal stresses at failure are related by the following equation:

$$\sigma_1/Q_u = \sigma_3/Q_u + \sqrt{m\,\sigma_3/Q_u + s}$$

Where σ_1 and σ_3 are the major and minor principal stresses at failure; Q_u is the uniaxial compressive strength of the intact rock material; m and s are constants that depend on the properties of the rock and on the extent to which it had been broken before being subjected to the failure stresses σ_1 and σ_3. Values of m and s are reported by Hoek and Brown for a wide range of rock types (7).

This empirical strength criterion has been used in conjunction with elastic stress analysis to determine the likely extent of over-stressed zones around underground excavations. This approach would be useful in the early stage of mine design when a mining method has been tentatively selected with little information available concerning the properties and behaviours of the rock mass but mine design decisions have to be made with regard to stope/pillar dimensions and proper sequences of mining. For this reason, it has been incorporated into the MINTAB program. However other failure criterion can be easily adopted by re-coding the two simple user-supplied routines if this empirical strength approach proves to be inadequate.

Based on the elastic stresses calculated from the MINTAB code, the major and minor principal stresses (σ_1 and σ_3) are then evaluated. Having determined the principal stresses at the centroids of each grid element (small square) the user-supplied routines will also calculate the available strength at those points in terms of the major principal stress at failure using the computed values of σ_3 and the given values of Q_u, m and s. The rock mass strength that can be developed for each grid element are then compared with the computed values of σ_1 (compression) to yield a series of strength/stress ratios which forms the contours. Potentially over-stressed zones around excavations can then be identified. If tensile stresses are present the point contour is identified as a negative number; the stability of these areas will be examined individualy based on available tensile strength data for the rock masses.

No experimental data on the strength of intact rock samples and the two constants defining empirical equations are available at this stage. The following values were assumed for this preliminary trial:

uniaxial compressive strength	75 Mpa
constant m	1.7
constant s	0.004

With the coefficients of m and s being 1.7 and 0.004 respectively, the rock mass is identified as a good quality rock with a CSIR rating of 65. (7). This input data can be modified when more information on the strength characteristics become available.

7. ANALYSIS OF RESULTS

7.1 CLOSURE/RIDE CONPONENTS

The rides (DX, DY) and closure (DZ) are defined as :

$$DX = U_x\ (Z_r = 0_) - U_x\ (Z_r = 0+)$$
$$DY = U_y\ (Z_r = 0_) - U_y\ (Z_r = 0+)$$
$$DZ = U_z\ (Z_r = 0_) - U_z\ (Z_r = 0+)$$

where displacements U_x, U_y and U_z are positive if they pointed in positive directions of the reef coordinate axies, X_r, Y_r, and Z_r, respectively (3). Z_r = 0+ and Z_r= 0 - represent the roof and the floor of the seam (orebody), respectively. A positive value of DZ indicates an excavation being closed or compression of the seam material (orebody).

The ride components along the stike and dip directions for the mining steps are shown in Figs 5 to 6 and Figs 15 to 16, for step 1 (existing stope) and step 2 (completion of primary stoping), respectively; the magnitudes of ride are very small and probably undetectable. The maximum closure is about 2.7 cm and occurred in the central part of 826.0 stope. The increase of closure as a result of completion of the primary stoping is not appreciable (from 2.73 cm to 2.79 cm). This is perhaps due to the fact that the volume of ore removed at the second step is relatively small.

The plots of closure for steps 1 and 2 are shown in Figs 7 and 17, respectively.

7.2 STRESSES

All six stress components in reef coordinates are presented in Figs 8 to 13 and Figs 18 to 23, for mining steps 1 and 2, respectively.

The pre-mining stresses at a depth of 310 m (top of model) are approximately 8.06 Mpa, 12.25 Mpa and 13.54 Mpa, respectively, for σ_{zz}, σ_{xx} and σ_{yy} in the global coordinate system. At the bottom of model where the depth is approximately 620 m stresses (in the same order and in the global coordinates) are 16.12 Mpa, 24.50 Mpa and 27.08 Mpa.

The calculated stresses in the reef coordinates for an element at a depth of 310 m (7th column. 3rd row from top) are 12.725 Mpa, 12.218 Mpa, and 9.303 Mpa respectivlely for SIGZZ, SIGXX and SIGYY. Transforming them into the global system, σ_{zz}, σ_{xx} and σ_{yy} are obtained as 8.369 Mpa 12.218 Mpa and 13.759 Mpa. This is fairly close to the assumed pre-mining stresses.

Similarily, the global stresses σ_{zz}, σ_{zz} deh σ_{yy} calaculated from a set of reef stresses for an element at a depth of approximately 620 m (7th column, 3rd row from bottom) are 16.357 Mpa, 24.409 Mpa and 27.253 Mpa, respectively. Again these stresses approach the assumed pre-mining stresses at that depth. This means that the influence of mining on stress in the upper and lower portions of the model is nominal.

Stresses around the stopes and in the pillars vary considerably as shown in Figs 8 to 13 (mining step 1) and Figs 18 to 23 (mining step 2). As expected, higher stresses appeared to be along the pillar edges and at the corners of some stopes. All the normal stresses (compression) are within the assumed uniaxial compressive strength. However, it is interesting to note that tensile stresses are observed in some areas of the hangingwall/footwall in the stopes. Rocks are less capable of sustaining tensile stresses; if fractures or joint sets are critically oriented in relation to those tensile stresses, these areas would be subject to instabillty. The lower portions of the stopes were already backfilled while the upper portions are still open; these areas should be inspected to see whether any instability is developing either by close examination or remote optical observation from a distant area.

7.3 STRENGTH/STRESS RATIO:

As discussed previously, an empirical strength failure criterion has been implemented within MINTAB program. The contours of the strength/stress ratio were calcuated based on the assumed strength parameters. They are shown in Figs 14 and 25, for the mining steps 1 and 2, respectively. No overstressed zone around the excavations has been developed. This indicates that the ground condition at this stage is stable if the assumed properties of rock mass and the in situ pre-mining stress field are representative of the 'true' condition of the mine. Obviously, any change of the coefficients relating the empirical strength equation and the compressive strength, plus a change in the pre-mining stress conditions, would modify these ratios considerably.

Nevertheless the strength/stress ratio is useful and provides some indication of the likely response of the rock mass to a proposed sequences of mining. It is particularly useful in this preliminary stages of design when a series of trial analysis may be required.

7.4 RELEASED ENERGY

The relationship between rock failures, strata control problems and the spatial rate of energy release has been discussed by Cook (8). It is stated that the energy release and the stored strain energy depend upon the volumetric convergence of the excavation. If the surfaces of excavation are stress free the released energy and stored strain energy are equal. If they make contact with one another, or with support loads then the surfaces of excavation are not free of stress and the released energy is greater than the stored strain energy. The released energy and stored strain energy can be reduced by filling an excavation with backfill only if the fill reduces the volumetric convergence of the excavation. Large differences in the rates of energy release can result from different ways of extracting the same volume of rock mass and this can be very significant in the excavation of highly stressed remnants where excessively high rates of energy release may cause severe problems or rock failure (9).

However, this concept as applied to the evaluation of underground structure stability has not been fully developed. Experiences and judgements will be required to assess the usefulness of these concepts in due course.

7.5 STRESS VARIATIONS ALONG 826.5 PILLAR

Pillar stresses along the two sections (AA and BB as shown in Fig. 4) of 826.5 pillar were calculated in both reef and global coordinate systems. Section AA is along the dip direction and is approximatley 6.85 m in from 825 stope face; section BB is also along the dip direction but located at the centre of the pillar (16 m in from 825 stope face).

The stress variations along the pillar resulting from the first mining step and the stress changes due to the completion of the 2nd mining step are shown in Figs 26 - 27, and Figs 28 - 29 for the reef and global coordinate systems respectively. These stresses are calculated at the mid-plane of the seams (orebody) i.e. $Z_r = 0$ in the mining horizon. It is reasonable to expect that the increase of stresses along the pillar edge resulting from further mining in the adjacent area would be greater as compared to that acting along the centre of pillar.

The vertical pre-mining stress in the global system is also plotted in Figs 28 and 29. It is seen that at the upper and lower levels where the influence of mining is nominal, the resultant stresses approach the pre-mining stress state.

8. CONCLUSIONS

Based on the results as discussed previousely the following remarks can be drawn:

(a) The displacement discontinuity technique does provide an inexpensive and very flexible means for evaluating stresses in a planar deposit with irregular and complex mining geometries; it is suitable for simulating mining layouts in order to assess mine stability in relation to mining method and mining sequences.

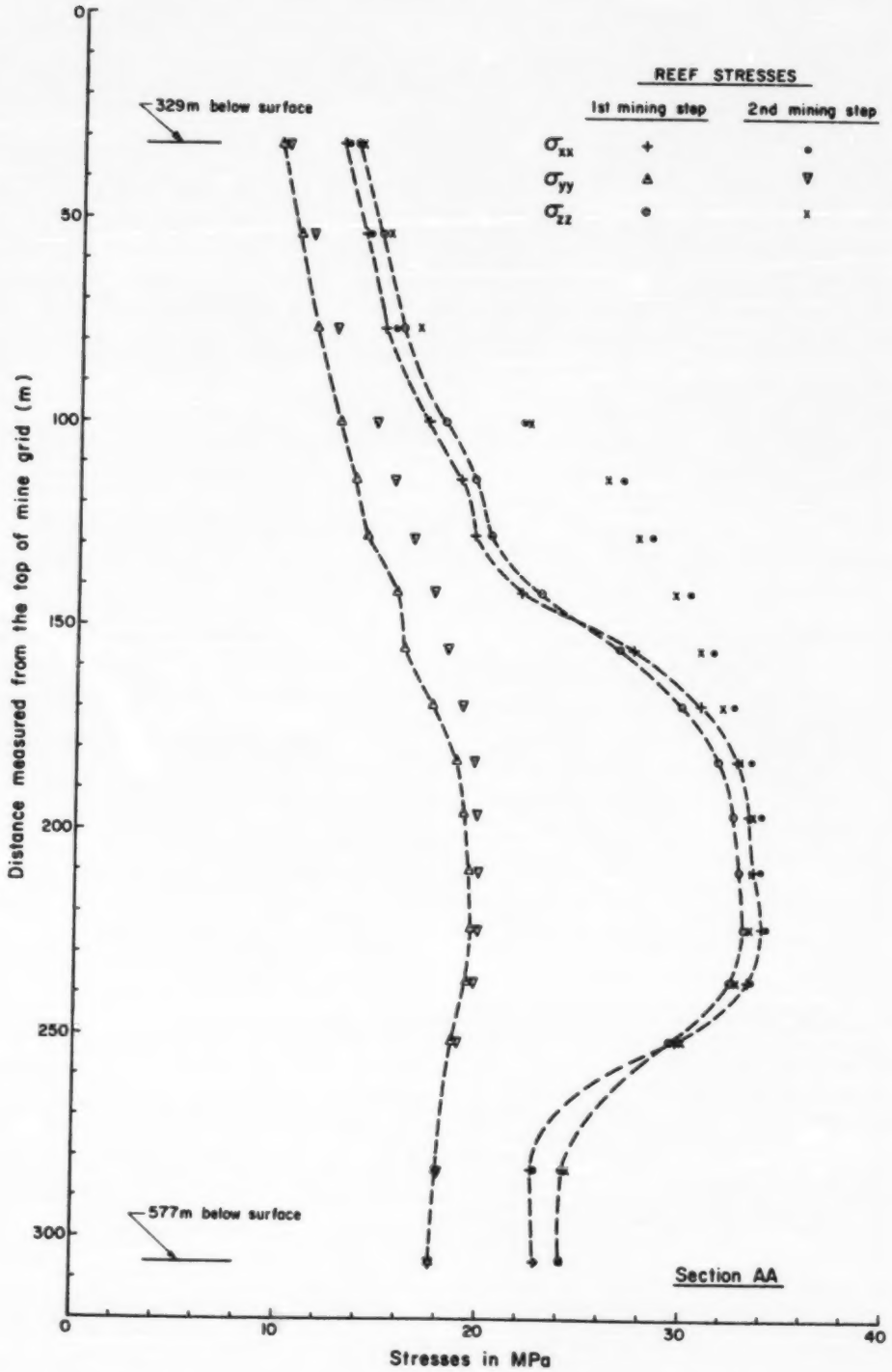


Fig. 26 - Stress variations (in reef coordinates) along 826.5 pillar, C.C.S. mine model

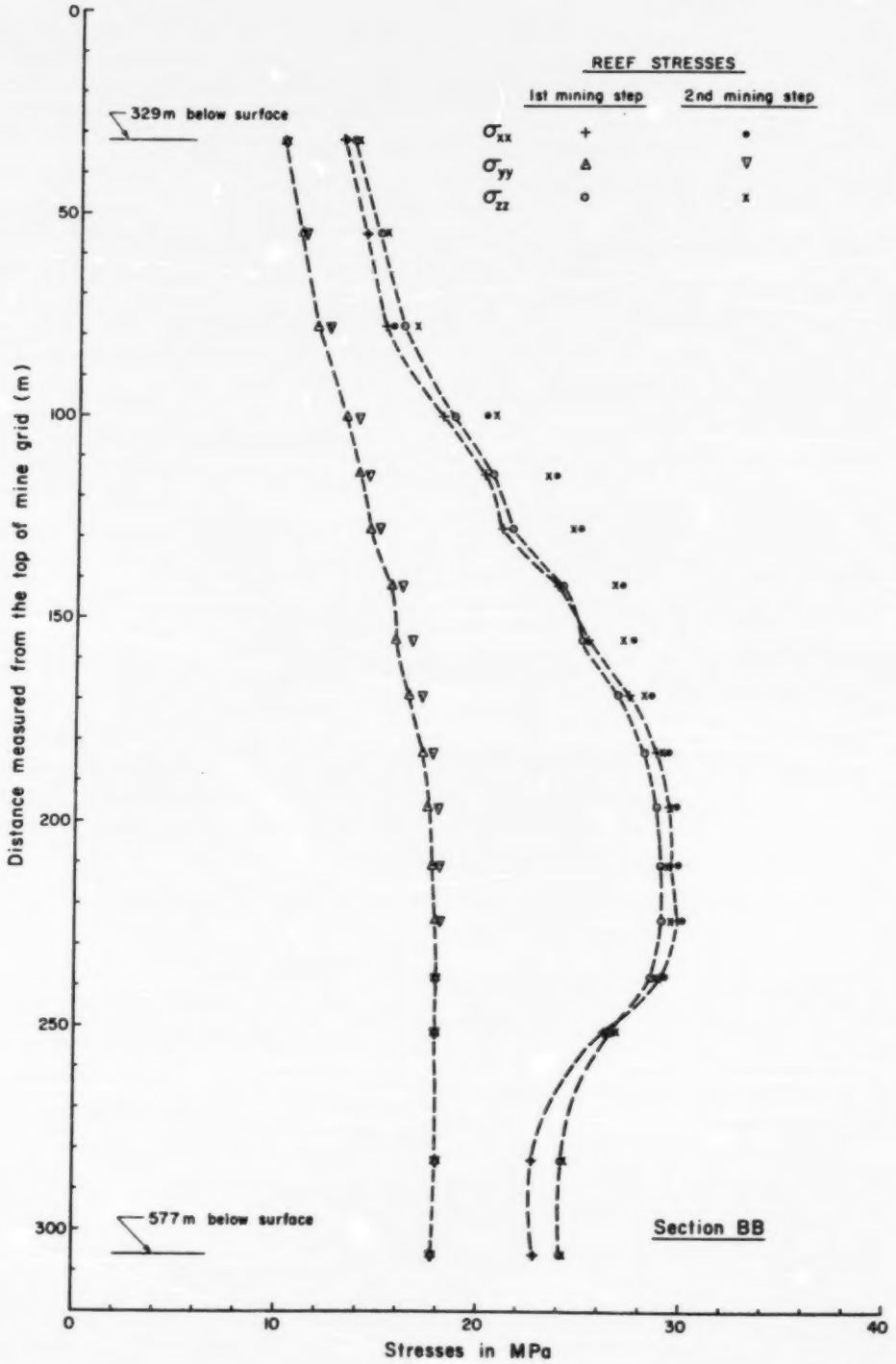


Fig. 27 - Stress variations (in reef coordinates) along the section BB of 826.5 pillar, C.C.S. mine model

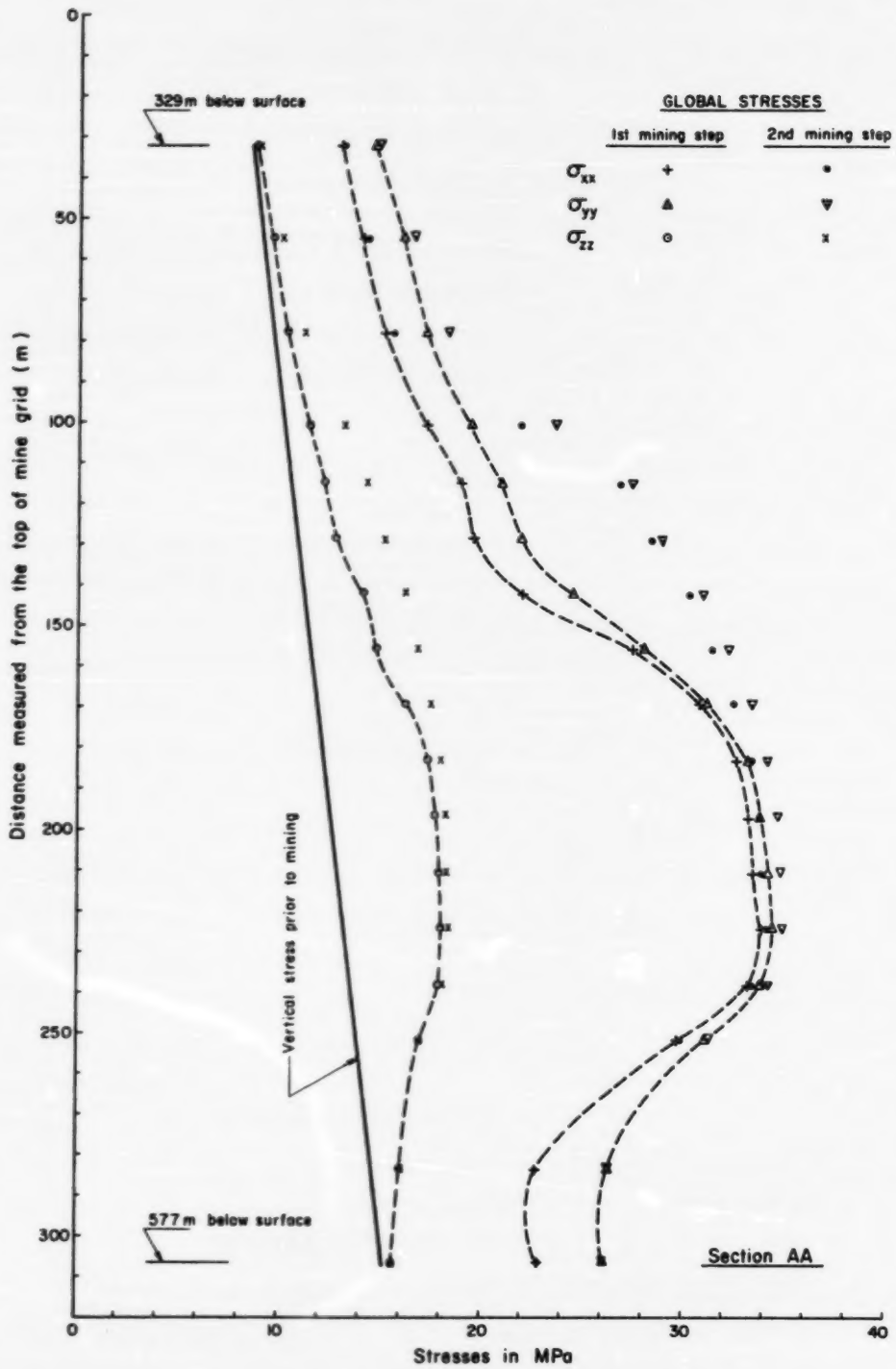


Fig. 28 - Stress variations (in global coordinates) along the section AA of 826.5 pillar, C.C.S. mine model

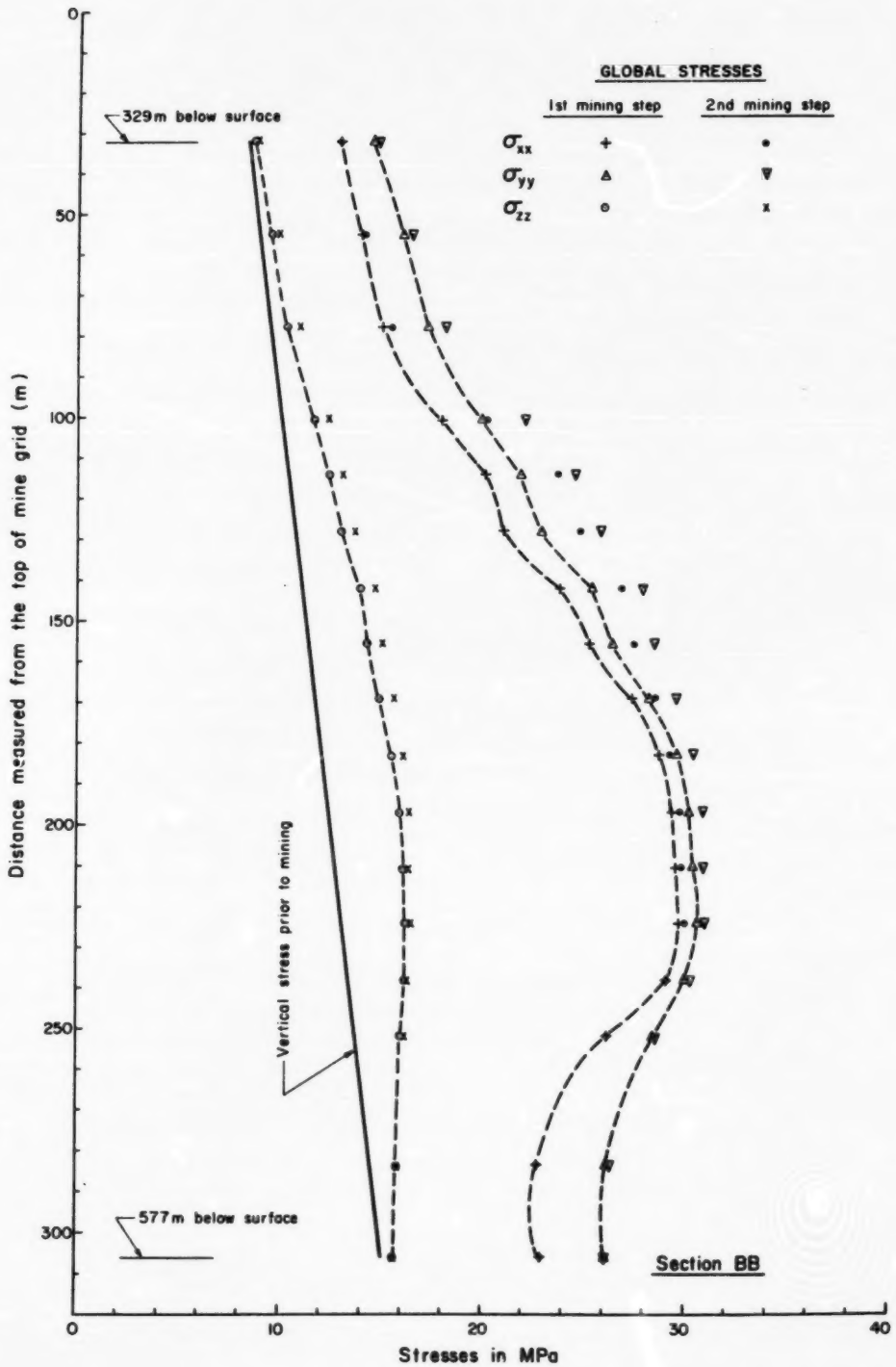


Fig. 29 - Stress variations (in global coordinates) along the section BB of 826.5 pillar, C.C.S. mine model

The results predicted from the C.C.S. mine model seem to be reasonable and very encouraging. However, the accuracy of results, the usefulness and validity of this approach as applied to blasthole stoping has to be verified by field measurements, monitoring and observations.

(b) If the assumed strength properties and pre-mining stresses are representative of the 'true' condition of the 830 orebody then the stresses developed from this mine model are within their elastic limits. They are not high enough, at this stage, to pose any instability problems. This is also indicated by the plots of strength/stress ratios which yield a minimum factor of approximately 1.6. However, this model should be re-checked when essential input parameters become available.

(c) Recent improvements to the MINTAB Program are the implementations of compressibility of the seam material and that of a modified procedure for stress/displacement calculations. The latter enables the stresses to be calculated anywhere within the rock mass and subsequently the pillar stresses acting along several sections of 826.5 pillar were evaluated.

(d) Backfilling is an integral part of the blasthole stoping operations at C.C.S. mine. Its response to the induced stresses whould be interesting in terms of ground control and therefore its effects on stresses/displacements should be examined. A backfill procedure has been incorporated within MINTAB and an initial testing on small model indicates satisfactory operation. A trial run of C.C.S. mine model with backfill will be carried out shortly.

(e) The sites to be selected for in situ stress (pre-mining) determinations should be located out of the influence zone of mining as far as possible. Preferrably these locations should be located above 1250 level and/or below 1750 level.

9. ACKNOWLEDGEMENT

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11. APPENDIX A - LIST OF SYMBOLS

The following 9 quantities are all in reef coordinate system:

DX	ride component in x-direction (parallel to strike)
DY	ride component in y-direction (parallet to dip)
DZ	vertical closure (in z-direction)
SIGXX	normal stress in x-direction
SIGYY	normal stress in y-direction
SIGZZ	normal stress in z-direction
SIGXY	shear stress in the xy plane
SIGYZ	shear stress in the yz plane
SIGZX	shear stress in the zx plane

The following 6 quantities are in global coordinates:

σ_{xx}	normal stress in x-direction (same as reef coordinate)
σ_{yy}	normal stress in y-direction
σ_{zz}	normal stress in z-direction
σ_{xy}	shear stress in xy plane
σ_{yz}	shear stress in yz plane
σ_{zx}	sheer stress in zx plane

Other quantities:

σ_1	major principal stress
σ_3	minor principal stress
Q_u	uni-axial compressive strength
m	constant
s	constant
S.S.R.	a strength/stress ratio